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UNITED NATIONS INTERREGIONAL SEMINAR ON THE APPLICATION OF ADVANCED MINING TECHNOLOGY

Proceedings

OTTAWA, CANADA

21 May-3 June 1973

UNITED NATIONS

DP/UN/INT-72-064

United Nations Development Programme

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New York, 1975

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CONTENTS

Page

INTRODUCTION 1

I. TONNAGE-GRADE ESTIMATION FOR MINERAL DEPOSITS AND ASSESSMENT OF ORE RESERVES (with tables 1-2, figures 1-7, bibliography)

B. W. Mackenzie and J. E. G. Schwellnus 4

II. EXPLOSIVES: PRODUCTS AND BLASTING IN CANADIAN MINES (with bibliography)

W. B. Morrey 35

III. MINE ENVIRONMENTAL ENGINEERING (with figures 8-14, bibliography)

M. J. McPherson 57

IV. ROCK MECHANICS AND GROUND CONTROL (with figures 15-44, bibliography)

D. G. F. Hedley and G. Herget 89

V. MINE SUPPORT (with table 3, figures 45-56, bibliography)

J. C. Wilson 130

VI. HARD-ROCK MINING IN METALLIFEROUS MINES (with figures 57-81)

Ingvar Janelid 161

VII. SINKING THE CREIGHTON NO. 9 SHAFT AT SUDBURY (with figures 82-83)

W. R. Dengler 203

VIII. UNDERGROUND MINE PLANNING AND DEVELOPMENT (with tables 4-8, appendix)

R. S. Haflidson and J. Toivanen 210

IX. TRANSPORT OF MINERALS

F. H. Skelding 231

X. DEVELOPMENT OF MINING IN BRITISH COLUMBIA (with tables 9-10, figures 84-87, bibliography)

J. B. Evans and J. S. Drake 253

/...

CONTENTS (continued)

Page

XI. OPEN-PIT EQUIPMENT SELECTION AND MAINTENANCE (with tables 11-17) 268
Frank H. Buchella, Jr. . . . .

XII. MATERIALS HANDLING RELATED TO MARINE BULK TERMINALS (with tables 18-20, figures 88-99, bibliography)
A. D. Fernie 292

Annexes

1. List of participants 330

2. Seminar Committee 335

Tables

(Mackenzie and Schwellnus, 1-2)

1. Tonnage and grade estimates: Maggie Canyon 25
2. Statistical analysis: Maggie Canyon manganese deposit 26

(Wilson)

3. Summary of data on support media 159

(Haflidson and Toivanen, 4-8)

4. Mining costs 216
5. Costs related to mode of entry 227
6. Preproduction and capital costs per ton amortized over four years . 228
7. Summary of preproduction and operating costs related to modes of entry 229
8. Summary of operating results: a hypothetical 3 million ton ore-body at various mining rates 230

(Evans and Drake, 9-10)

9. Relative importance of mining to British Columbia 263
10. Active mining operations in British Columbia 266

/...

Tables (continued)

Page

(Buchella, 11-17)

11. Site preparation equipment 269
12. Loading equipment 272
13. Haulage equipment 275
14. Field support and service equipment 280
15. Equipment selection chart 285
16. Open-pit mining equipment 287
17. Mining costs by function expressed as a percentage of the total direct cost 290

(Fernie, 18-20)

18. Determination of berth occupancy (example) 316
19. Examples of actual loading times 318
20. Travelling versus fixed shiploader rates 321

Figures

(Mackenzie and Schwellnus, 1-7)

1. Mineral deposit tonnage-grade classification 6
2. Sample frequency distribution 10
3. Grade trends 14
4. The lognormal distribution 15
5. Exponential tonnage-grade relationship 17
6. Prophyry-copper mine development: sensitivity of profitability to change in input variables 19
7. The economic sampling limit 22

(McPherson, 8-14)

8. Effect of air velocity on pick-up of dust 60
9. Influence of pick speed and depth of cut to airborne dust 61
10. Duct systems for auxiliary ventilation of tunnels 64
11. Cyclic pattern of airflow and gas emission 67

/...

Figures (continued)

Page

12. Sorption isotherms for dry coals (Mücke) 69

13. Mine refrigeration and water reticulation system 75

14. Information flow chart for ventilation planning 82

(Hedley and Herget, 15-44)

15. Rock mass, a composite of rock material and structural defects . . . 91

16. Stress-strain curve for quartzite 93

17. Distribution of elastic modulus and uniaxial compressive strength . 93

18. Relationship between consistency and unconfirmed compressive strength 95

19. Example for describing geological setting of a mine area 97

20. Overcoring technique and example of measurements with the deformation meter 100

21. Geometry and test equipment of C.S.I.R. strain cell (doorstopper) overcoring technique 101

22. Geometry for C.S.I.R. triaxial strain cell overcoring technique . . 102

23. Increase of vertical stress with depth 103

24. Increase of average horizontal stress with depth 105

25. Stress trajectories from finite element models for k = 0.33 and k = 3 107

26. Distribution of pressure on elastic and yielding pillars 107

27. Probability of failure for pillar in sound siderite calculated from in situ stresses and laboratory strength tests on samples of different size 109

(28-29. Numbers not used)

30. Layout of anchors and wires in a borehole 111

31. Example of extensometer measurements in a pillar 111

32. Horizontal movement between 1932 and 1951 in vicinity of the San Andreas fault 113

33. Contours of seismic velocity before and after destressing 114

34. Movement of a pit slope leading to failure 114

35. Observed pillar loads as a function of calculated pillar strengths for collapsed and stable areas of mining 117

/...

Figures (continued)

Page

36. Location of uranium mines in the Elliot Lake area 118
37. Typical layout of stopes and pillars in the Elliot Lake mines . . . 118
38. Estimated pillar stresses and strengths, Elliot Lake 119
39. Variation of pillar width and extraction with depth 119
40. Design and layout of cable bolts 121
41. Variation of load on cable bolts 121
42. Layout of transverse cut-and-fill stopes 123
43. Layout of "post" pillar mining method. 123
44. Design nomograph for "post" pillars 125

(Wilson, 45-56)

45. Geological map of Sudbury Basin showing operating mines 131
46. Strathcona mine: fill preparation, plant flowsheet 136
47. Photo of free-standing cement: tailing without fill fences 139
48. Plan of Onaping West orebody: below 1650 L 140
49. Undercut and fill, showing timber support 141
50. Diagrammatic layout for post-pillar cut-and-fill stoping 142
51. Gangway sets 146
52. Undercut-and-fill, showing timber 147
53. Heavy-duty three-piece yielding arches 149
54. Rock-bolts, complete 150
55. Tensioned cable bolts 153
56. Application of Weldmesh 155

(Janelid, 57-81)

57. Development programme for drill rigs 164
58. Drilling pattern, benching with wide spacing 167
59. Approximate costs per cubic metre when hard rock is disintegrated . 169
60. Loading and hauling capacity 170
61. Inclined transport, Tuolluvaara 176
62. Long-hole mining, Stråssa 177

/...

Figures (continued)

Page

63. Sublevel stoping, Vintjärn 178
64. Sublevel stoping, Idkerberget 180
65. Fill material 182
66. The use of various filling methods 183
67. Drift-and-fill mining 184
68. Undercut-and-fill: Kristineberg mine 185
69. Development of the sublevel caving method 187
70. Fundamental parameters for sublevel caving 188
71. Volumes of motion at sublevel caving 190
72. Back break, after blasting a round at sublevel caving 191
73. Gravity flow curves, maximum height, width and depth of volumes of motion: from full-scale and model tests 192
74. Longitudinal mining: drift situated in ore 193
75. Longitudinal mining: drift situated partly in foot-wall 194
76. Drilling pattern, comparing sublevel caving and front caving 196
77. Block caving, Grängesberg 197
78. Block caving, El Teniente 198
79. Induced block caving, Russia: drilling and blasting against already caved ore, 1964 and 1972 200
80. Block caving, Creighton Mine 201
81. Ventilation principles in drifting 202

(Dengler, 82-83)

82. General arrangement of Creighton No. 9 shaft 205
83. Plan of Creighton No. 9 shaft 205

(Evans and Drake, 84-87)

84. Generalized physiographic regions of British Columbia 255
85. Mineral production in British Columbia, 1900-1972 257
86. Volume of mineral production in British Columbia, 1900-1972 258
87. Reported value of British Columbia mineral production, 1900-1972 . . 262

/...

Figures (continued)

Page

(Fernie, 88-99)

88. Large storage and reclaim system 293

89. Causeway, dock and shiploader: Port of Chaguaya, Lake Titicaca, Bolivia 295

90. Gradall bucket: rail-car unloading 298

91. Side dump rail-car unloader 299

92. Rotary car dumper: Pacific Coast Bulk Terminals Ltd., British Columbia 301

93. Storage pile: cone and multi-cone 305

94. Storage pile: kidney and rectangular tent-shape 306

95. Storage pile: tent and flat-top rectangular 307

96. Rotary plows and portal scraper 312

97. Fixed buffing loader: Skagway ore terminal, Alaska 319

98. Concrete pontoon and shiploader: Jedway Iron Ore Ltd., British Columbia 322

99. Travelling/floating buffing and shuttling loader: Pacific Coast Bulk Terminals Ltd., British Columbia 323

/...

INTRODUCTION

The Economic and Social Council in its resolution 345 (XII) of 9 March 1951 recognized the importance of promoting the effective and sustained use of the world's natural resources as a means of furthering economic development, and recognizing further the progress that may be made in this direction through the holding of international conferences in which there is an exchange of information pertinent to particular types of resources or resource problems common to a group of countries. Accordingly, the Secretary-General was requested to explore, at the request of Member States concerned, and, when appropriate, in consultation with regional economic commissions and specialized agencies, the scope and specific subject-matter of resource conferences that may, upon investigation, appear desirable.

It was with this purpose that the United Nations organized the Interregional Seminar on Advanced Mining Technology. The Seminar, which was jointly financed by the United Nations Development Programme and the Government of Canada, was held in Ottawa and in the mining camps of northern Ontario from 21 May-3 June 1973. It was attended by 34 participants from developing countries who were senior government or industrial officials concerned with the development and modernization of mining operations (see annex I). A number of experts delivered lectures (annex II). As noted above field visits were organized to a number of the most technologically advanced mines in northern Ontario, both open-pit and underground.

Mr. Jean-Paul Drolet, Assistant Deputy Minister of the Department of Energy, Mines and Resources, welcomed the participants on behalf of the host Government.

The Chairman of the Seminar Committee, Dr. John Convey, Director, Mines Branch, Department of Energy, Mines and Resources, in his speech at the opening of the Seminar pointed out the complexities involved in today's mining operations. The first of these is the need to combat steadily rising unit costs and at the same time to meet consumers' needs by constantly having to move much greater quantities of ore more efficiently as average grades decline and consumption rises. This has to be done while facing the restrictions placed upon mining in recent years by standards aimed at preventing pollution.

The Seminar, of more than considerable value in itself, should have decidedly increased impact as part of a continuous process. For two decades, the Department of Economic and Social Affairs of the United Nations has maintained a programme of seminars and workshops dealing with the earth sciences aimed at co-ordinated and sustained transfer of technological knowledge among nations. As part of the process the meeting in Ottawa was planned to precede the Interregional Workshop on Negotiation and Drafting of Mining Development Agreements which met in Buenos Aires about six months later, in November 1973.

The main thrust of the successive meetings may appear to be widely separated, one highly technical and the other basically legal with political, economic and sociological overtones. In actual fact, the two must be inextricably connected. They constitute the prime movers for new mining developments, transcending in importance even geological factors. Current world shortages in regard to a number of basic mineral needs only rarely reflect lack of exploitable deposits either known or susceptible to relatively easy discovery. Instead, the fundamental necessity today for continuing creation of new mining capacity in harmony with world needs is the marriage of technology and capital within political environments that give full consideration to the risks and complexities of the minerals industry.

In a world characterized by imbalances, one of the more serious is the concentration of mining activities in a very few countries. Outside of the Centrally Planned Economies, two thirds of the world's major metallic mines are found in four countries - the United States of America, Canada, Australia and South Africa. Geology not being politically controlled, one has to consider the factors which have led to such concentration. By and large, the infrastructure is well developed although mines in northern Canada and western Australia frequently have had to create all or most of transportation, power, housing and maintenance needs. Manpower pools in regard to technical, financial and management skills are not only ample but reinforced also by a great deal of interchange between these countries which share a common language. However, people can be trained and infrastructure built, so the key element in development of the mining industries of the nations cited above is that there has been no lack of development capital, except in the short term. Traditionally, in the youthful stage of mining said capital is largely foreign in origin. To illustrate, it is frequently overlooked that in the nineteenth century, European money was the prime mover in the growth of American mining.

The Seminar was designed to make a broad-brush review of the latest developments in mining technology. The history of the mining industry during this century has seen output increasing exponentially while grades of ore and selling price, in terms of constant dollars, have generally decreased. It is obvious that the key has been technological advances. Yet these have rarely been truly revolutionary. The most momentous event in modern mining, the introduction in 1907 of mass extraction of low-grade (for that day) copper ore at Bingham Canyon, Utah, was, in essence, the continuation of a process which began when the first prehistoric miner was joined by a second. Possibly the invention of flotation in the first quarter of this century could be viewed as being completely original. Almost all other great steps forward have involved simply the further development of existing concepts, modification of methods, improvement in equipment, refinement of procedures and the arts of management. The break-throughs which can be seen in the near future are of a similar nature such as nuclear explosions for in situ leaching, complete automation of processing plants, extraction of deep-sea nodules, great expansion in the production of geothermal energy, the construction of breeder reactors and the harnessing of nuclear fusion.

It is proposed that the advance of technology is not dependent upon luck or the brilliant idea which turns the world upside down. There is, thus, no basis for

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believing that the wells of technical achievement will dry up. The mental and mechanical acts of this century that led to the rise in productivity of the American miner, for example, on a super-exponential curve should not be viewed as short-lived phenomena. Indeed, it would be difficult to make a case that technology is finite. But it is costly. Therefore, a basic problem in many developing countries is not only to keep abreast of evolving techniques but also to create policies and mechanisms for obtaining the capital required for implementation.

The first week of the Seminar was devoted to presentation and discussion of the papers listed under Contents. It was manifestly beyond the grasp of any individual, no matter how well prepared, to absorb more than a small part of the whole. Accordingly, the second week which was spent visiting some of the most modern mines of the world was of fundamental importance in giving visual expression to so much of what came out in the papers. The field trip took place in northern Ontario and involved:

1. An underground tour of the Denison uranium mine and the Mining Research Centre of the Department of Energy, Mines and Resources, both at Elliot Lake.
2. Underground tour of the Creighton Mine and the Strathcona Mine, both in the Sudbury Basin.
3. Underground tour of the Ecstall copper-lead-zinc mine near Timmins.
4. Tour of open-pit Adams iron ore mine near Kirkland Lake.
5. Visit to the School of Mines, Haileybury.
6. Tour of open-pit Sherman iron ore mine, Timagami.

Finally, it is strongly suggested that the principal obstacle to the transfer of technology is not a function of brainpower. Many developing countries have net wastage of such because the technical people do not have the tools with which to work. It cannot be over-emphasized that the primary necessity is the marriage of technology and capital referred to above. Given this, the skills required for effective industrial growth can be found and/or developed, initially to a considerable degree from abroad, eventually from national ranks.

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I. TONNAGE-GRADE ESTIMATION FOR MINERAL DEPOSITS AND ASSESSMENT OF ORE RESERVES

B. W. Mackenzie and J. E. G. Schwellnus*

This paper reviews statistical and economic principles relating to the estimation of tonnage and grade for mineral deposits and the assessment of ore reserves. Emphasis is placed on developing a broad appreciation of the topic in terms of information requirements, techniques of analysis and application of results. Each mineral deposit is geologically unique and, thus, has special requirements for estimating tonnage and grade parameters. This paper focuses on fundamental concepts which are generally applicable rather than on detailed practice for specific deposit conditions. 1/

Mine development and mine operating decision problems differ from those of other industrial situations because of the conditions imposed by the geological environment, particularly the mineral deposit conditions. The fixed size of mineral deposits results, inevitably, in their exhaustion and affects the determination of mine capacity. Variations in grade within mineral deposits give rise to cut-off grade decisions. These mineral deposit conditions are determined by the estimation of tonnage and grade and the assessment of ore reserves.

A. Definition of ore reserves

Standard practices have been developed for the definition of ore and the classification of ore reserves. "Ore" is usually defined as a natural aggregation of minerals, the tonnage and grade of which make extraction commercially profitable. The classification of ore reserves - typically into "proven", "probable" and "possible" categories - is based on the amount and type of sampling carried out and on the distances over which sample values are projected. Such traditional definitions have been the subject of much debate for many years. This reflects the difficulty of developing standards that are simple, practical and generally applicable over a wide range of mineral deposit conditions.

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1/ A large number of case studies have been published, including those in the proceedings volume of the CIM conference on ore-reserve estimation and grade control (15).

The definition of ore reserves for estimating purposes should embody two distinct dimensions - a geological dimension reflecting the level and reliability of sampling information, and an economic dimension reflecting the relationship which exists between cost and revenue. A tonnage-grade classification for mineral deposits based on these properties is shown in figure 1. 2/

The level of sampling information is a function of the amount of exploration and development work which has been completed. The unknown part of the mineral deposit exists but has yet to be discovered. The known part of the deposit, including past production, has been discovered and at least partially delineated. Tonnage and grade parameters for the known part of the deposit may be estimated with differing degrees of reliability, depending on the level of sampling information.

The economic dimension subdivides the mineral deposit into economic and uneconomic categories based on the relationship between cost and revenue in different parts of the deposit. The economic margin is reflected in cut-off grade which is (at least conceptually) determined by the equation of marginal cost and marginal revenue. 3/

Thus, ore reserves are defined as the known, economic, unexploited part of the mineral deposit. Tonnage and grade estimates for ore reserves are classified - for example, into proven, probable and possible categories - to reflect differing reliability and confidence of estimates resulting from differing levels of sampling information.

A tonnage-grade classification, as shown in figure 1, is fixed for only a particular point in time. Changing geological and economic conditions over time shift the ore reserve boundaries. Mine production exhausts ore reserves. Exploration results in the discovery and delineation of unknown economic resources and the "proving-up" of known ore reserves.

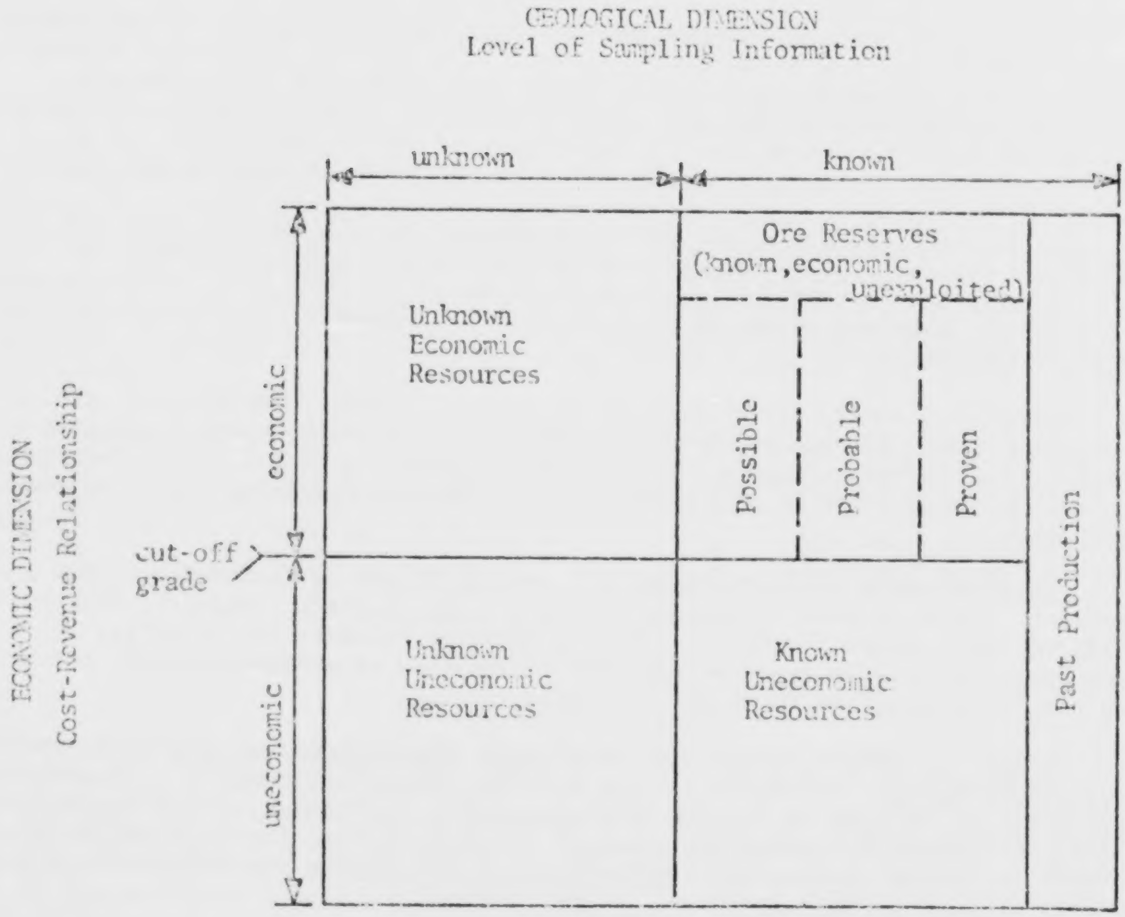
Changes in revenue result from short-term fluctuations and long-term trends in supply-demand-price conditions for the mineral commodities produced. Changes in cost occur as the mine moves from development into production, and with advances in mining and processing technology, changes in productivity, depth and depletion. Favourable changes in the cost-revenue relationship reduce cut-off grade and thereby convert known uneconomic resources to ore reserves.

Favourable changes in both geological and economic conditions are required for the conversion of unknown uneconomic resources to ore reserves.

2/ Comprehensive treatments of this subject are given by Brooks (2) and Zwartendyk (14).

3/ Practically, owing to uncertainties and changing economic conditions over time, cut-off grade is not usually directly determinable, particularly for mine-development decision purposes.

Figure 1. Mineral deposit tonnage-grade classification


GEOLOGICAL DIMENSION
Level of Sampling Information
unknown
known
ECONOMIC DIMENSION
Cost-Revenue Relationship
economic
uneconomic
cut-off grade
Unknown Economic Resources
Ore Reserves (known, economic, unexploited)
Possible
Probable
Proven
Past Production
Unknown Uneconomic Resources
Known Uneconomic Resources

Ore reserve definition must be dynamic, reflecting the cumulative effect of changes in geological and economic conditions over time. The two-dimensional concept of ore reserves outlined above provides guidelines for the estimation of ore reserves.

Ore reserve estimates are required to assist with the analysis of mine development and mine operating decisions. Such decisions concern choice of the optimum economic alternative from a number of technically feasible alternatives. To determine the optimum economic alternative, each technically feasible alternative must be evaluated. Decision parameters usually include selection of the optimum cut-off grade and ore reserves, and, therefore, ore reserve estimates are required for each alternative considered. These estimates are facilitated if, initially, over-all geological estimates of tonnage and grade for known resources are made. These geological estimates establish a base for the assessment of ore reserves for each technically feasible alternative. The specific conditions assumed for each alternative are imposed on the geological estimates to assess the tonnage and grade characteristics of ore reserves. Geological estimates are only a function of the level of information available and, thus, are fundamental to assessments which embody differing sets of economic conditions.

The tonnage and grade characteristics of ore reserves are initially estimated in terms of content within the mineral deposit. For use in economic evaluation, such estimates must be converted to a "recoverable" basis to assess the tonnages and grades which will actually be mined and recovered. This requires estimates of mining and milling recovery, and dilution, based on knowledge of the geological environment, and the mining and processing methods to be employed.

B. Geological environment

The mode of origin of a mineral deposit influences its tonnage and grade characteristics, and therefore, should have an important bearing on sampling strategy and the application of techniques for tonnage and grade estimation.

The occurrence of mineral deposits is controlled by the geological environment. Mineral deposits can be host- or wall-rock controlled, structurally controlled, or climatically and topographically controlled. Most deposits are formed as the result of combinations of these factors. Usually the host-rock provides the regional constraint for mineralization, and structural features the local constraint. Thus, mineralization is not usually random but rather has regional trends and local variations depending on these geological features.

The tonnage and grade of ore reserves is assessed by imposing an assumed cut-off grade on over-all geological estimates. The significance of variations in cut-off depends on geological features of the mineral deposit, particularly the relationship which exists between tonnage and grade. For this purpose, Lacy (7) has classified mineral deposits into three categories:

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1. High-grade, relatively small- to moderate-sized deposits with high unit value and generally a wide margin of operating profit. This type of deposit is normally worked with a low capital investment and small ore reserves. It is common for a sharp boundary to exist between ore and waste; ore reserves are high-grade and adjacent material contains essentially no values. Variations in cut-off grade will normally have little effect on ore reserves and mine life as grades within the deposit will be considerably above the economic cut-off grade. This is reflected in the high profit margins which characterize the mining of this type of deposit. Tonnage rather than grade is usually the critical decision variable because the deposit limits are erratic and hard to define, or reserves are small, and economic criteria for investment are sensitive to mine life and mine capacity.

2. Low-grade bulk deposits of a size and shape that lend themselves to low-cost bulk-mining methods. This type of deposit is mined with a low profit margin and a high capital investment, based on high manpower productivity and large ore reserves. These deposits are characterized by a dispersed distribution of mineral values. Ore boundaries are determined through careful analysis of contained values. The shape and size of ore reserves will be altered considerably by minor fluctuations in cut-off grade. Grade rather than tonnage will be the critical decision variable. Grade trends within the deposit will be important because they may present an opportunity for mining above the average grade of ore reserves in the early years of mine life. This has an important influence on economic criteria for investment.

3. Disseminated deposits with dispersed values, having a relatively small and concentrated enriched zone capping the deposit. The enriched zone has the characteristics of the high-grade type of deposit while the unenriched primary zone is similar to the low-grade bulk deposits. Variations in cut-off grade will have little effect on mining the enriched zone, but economic and technological conditions which lower the cut-off grade to concentrations contained in the primary zone will result in a large increase in ore reserves. Porphyry copper mines in the south-western United States provide a good example of this type of deposit. In the early development of the porphyry coppers only the blanket of enriched ore capping the top of the deposit could be profitably mined. Values in the enriched blanket averaged 1-1.5 per cent copper. It was not until economic and technological conditions were such that cut-off grade dropped below 0.7 per cent copper that these deposits underwent tremendous expansion as ore boundaries extended into the unenriched material.

C. Statistical analysis for tonnage and grade estimation

Statistical techniques find practical application in the analysis of sample values for purposes of estimating tonnage and grade parameters. Statistical analysis also provides a conceptual insight into mineral deposit characteristics, estimating techniques, and the application of results.

Relevant deposit characteristics include all attributes of the deposit which influence the estimation of tonnage and grade, for example, individual mineral and metal assays, specific gravity and thickness. The natural variability of each characteristic is defined by the probability distribution of its population and is estimated by its sample frequency distribution. Estimating errors are a function of the degree of variability but can be controlled by the sample size.

Preparation of a sample frequency distribution involves dividing the sample values into class intervals and plotting the number of occurrences (frequency) as a function of the class interval as shown in figure 2. Sample frequency distributions, constructed for all relevant deposit characteristics, provide the starting point for the estimation of tonnage and grade parameters.

The shape of a sample frequency distribution is not fixed but will vary depending on the sample size selected. A small sample unit will reflect a larger variation between sample values than will a larger sample unit. However, when a sampling unit has been selected, the parameters of the probability distribution of the population being sampled are fixed.

The two most important parameters of the probability distributions for relevant deposit characteristics are the mean (expected) value and the standard deviation. Estimates of these parameters are made from their sample frequency distributions.

The mean value estimate ($\overline{X}$) may be determined by adding all sample values and dividing by the sample size (N):

$$\overline{X} = \frac{\Sigma X}{N}$$

Using the sample frequency distribution, a close approximation of the mean value is determined by multiplying the midpoint of each class interval (mp) by the frequency of occurrence in the class interval (f), taking the summation of these products for all class intervals, and dividing by the sample size:

$$\overline{X} = \frac{\Sigma\left[(f)\quad(mp)\right]}{N}$$

The reliability of the mean value estimate is a function of both the degree of variability of the deposit characteristic and the sample size.

Variability is estimated by the standard deviation of the sample frequency distribution. The standard deviation is defined as the square root of the mean of the squares of the deviations of values (d) about the mean value. The standard deviation of the sample (S) is an estimator of the standard deviation of the population. Thus:

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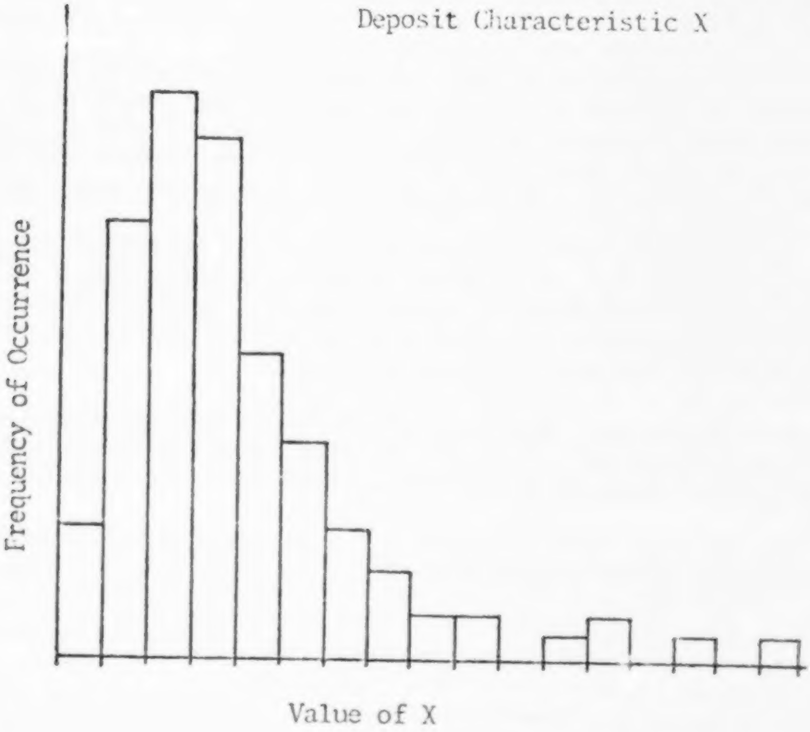
Figure 2. Sample frequency distribution



$$S = \sqrt{\frac{\Sigma(X - \overline{X})^2}{N}} = \sqrt{\frac{\Sigma\ d^2}{N}}$$

Using the sample frequency distribution, S is approximated by the following expression:

$$S = \sqrt{\frac{\Sigma f\ (mp)^2}{N} - \left[\frac{\Sigma mp}{N}\right]^2}$$

The standard deviation estimate is used to assess the reliability of the mean value estimate. The reliability of the mean value estimate is measured by the "standard error of the mean". [4]

$$S_{\overline{x}} = \frac{S}{\sqrt{N - 1}}$$

where,

$S_{\overline{x}}$ = standard error of the mean,

S = standard deviation of the sample frequency distribution,

N = sample size.

Thus, the standard error of the mean varies directly as the variability of a deposit characteristic as reflected by its estimated standard deviation, and inversely as the square root of the sample size.

The standard error of the mean defines the possible distribution of actual mean values about the mean value estimate. For reasonably large sample sizes, this mean value distribution will approximate the normal distribution, even if the sample frequency distribution is skewed.

The standard error of the mean is used to estimate confidence limits for the mean value estimate. These limits reflect the reliability of sampling and the risk or uncertainty associated with the mean value estimate.

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[4] Providing the sample data are random. Randomness assumptions and tests are discussed below.

/...

The mean and standard error of the mean estimates for tonnage and grade parameters are used to determine the economic sampling limit and to derive ore reserve estimates for the economic evaluation of alternatives.

The use of statistical analysis, as outlined above, requires that the sample data are random, that is, individual sample values are essentially independent of one another. Random data is a fundamental requirement for determining the reliability of sampling results and expressing reliability as confidence limits about the mean or expected value estimate. For data that are not random, there is no known means of quantitatively expressing reliability or confidence.

Hazen describes a number of statistical tests which may be applied to sample data to determine whether or not the randomness assumption is met (5). Nearest neighbour, equal expectancy and poisson distribution tests may be used to assess the randomness of drill-hole locations. Generally, systematic drill-hole patterns satisfy randomness criteria. The mean-square-successive-difference test is used to assess the randomness of successive sample values within a drill-hole. Randomness in this third dimension may be controlled by adjusting the sample size used, for example, there will be greater independence or randomness between successive sample values if 10-foot sample lengths are used as compared to 5-foot lengths.

Grade trends within a mineral deposit are non-random and complicate the estimation of grade parameters. Trend is the change in mean grade which occurs over some distance or some area which has been sampled. Trend is caused by the host-rock and structural features of the geological environment.

Trend may be a sharp change in grade across a structural boundary, or it may be a gradual change, either increasing or decreasing in assay value. However, the change must be persistent or continuous over some distance in contrast to the rapid or erratic changes over short distances which are associated with random variation.

All deposits have trend; that is, the grade changes either abruptly or gradually from uneconomic mineralization to high-grade ore. Grade trends within a deposit do not alter over-all deposit parameters, but these within-deposit characteristics influence subpopulation parameters, such as the grade of a high-grade zone within the deposit. An example of grade trends is shown in figure 3.

The best approach for sampling under conditions where trend is present is to subdivide or stratify the total deposit into zones of essentially no trend. The manner in which a deposit is stratified should be based on a knowledge of the geological environment as outlined in the preceding section. For many deposits grade is sufficiently uniform over large volumes that blocks may be stratified according to grade and assay data can be used to estimate grade parameters for these stratified blocks of substantially uniform grade or no trend.

As discussed in the preceding section, grade is the critical decision variable for most deposits of economic significance. Grade parameters are estimated from their assay frequency distributions. The mathematical representation (fitting

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of a curve) to an assay frequency distribution is of practical value when knowledge of the population must be inferred from the sample distribution.

It is generally recognized that frequency distributions of low-grade assays are positively skewed, medium-grade assay frequency distributions are normally distributed, and high-grade assay frequency distributions are negatively skewed. The majority of assay distributions are low-grade and positively skewed. The lognormal distribution, an example of which is shown in figure 4, provides a useful approximation of low-grade assay distributions. For a lognormal distribution, the assay values are not in themselves normally distributed, but if grade intervals are expressed in terms of logarithms, a frequency distribution of the logarithm values of assays within the logarithmic grade intervals becomes a normal distribution. Such a transformation is useful because there is a large amount of statistical theory and test information developed on the basis of the normal curve.

Assay frequency distributions, based on a fixed sample size, also provide an estimate of the tonnage-grade distribution within a mineral deposit. Given an over-all estimate of deposit size, assay frequencies may be applied to estimate the tonnage within each grade interval. Knowledge of the tonnage-grade relationship is useful when specific sets of economic conditions are imposed on geological assessments of tonnage and grade to derive estimates of ore reserves.

If attention is focused on grade estimates above some economically significant level of mineralization, the exponential distribution may be used to estimate the tonnage-grade relationship within a deposit. Using the exponential tonnage-grade relationship, as formulated and tested by Lasky (8) and Musgrove (12), the tonnage of ore increases geometrically as the grade declines arithmetically. Figure 5 shows the exponential distribution of ore reserves and metal content as functions of cut-off grade and average grade of ore reserves. These curves illustrate the following exponential relationships:

$$R(G) = Ae^{-G/K}$$

$$M(G) = KAe^{-G/K}(1 + G/K)$$

$$Gavg(G) = M(G)/R(G) = K + G$$

where G = cut-off grade;

$R(G)$ = tonnage of ore reserves above cut-off grade G;

$M(G)$ = mineral content in ore reserve above cut-off grade G;

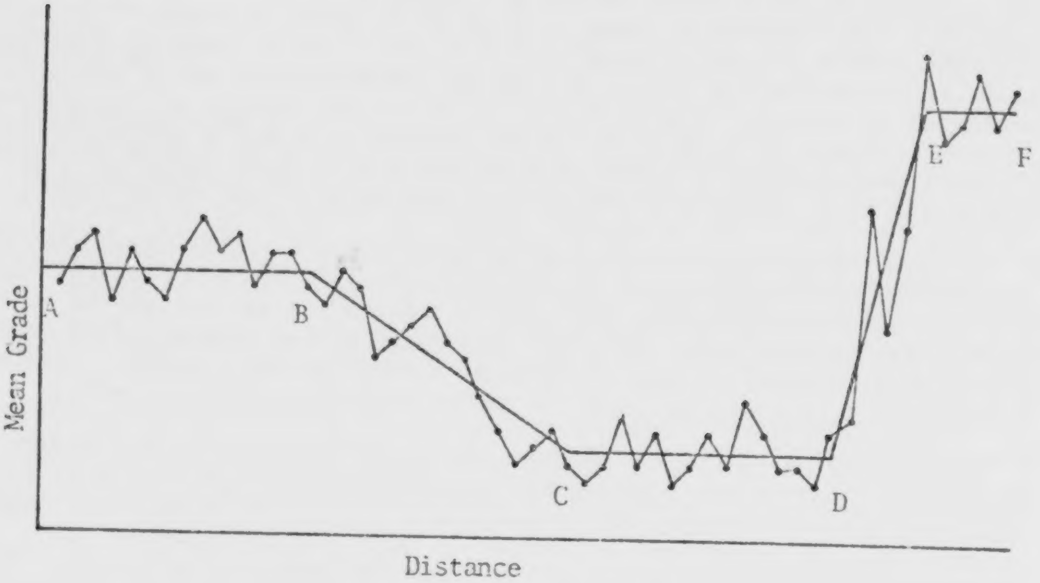
$Gavg(G)$ = average grade of ore reserves above cut-off grade G;

A = total deposit tonnage, i.e. $R(0) = A$;

K = average grade of total deposit, i.e. $M(0)/R(0) = KA/A$.

The following table illustrates the application of the exponential tonnage-grade relationship for a hypothetical copper deposit containing 200 million tons (a) with an average grade of 0.5 per cent copper (K).

Figure 3. Grade trends



sample assays, plotted by location

A-B
C-D } no trend, random fluctuations of grade about mean
E-F

B-C trend, gradual decrease in grade accompanied by random fluctuations in grade about decreasing grade trend.

D-E trend, abrupt increase in grade accompanied by random fluctuations in grade about increasing grade trend.

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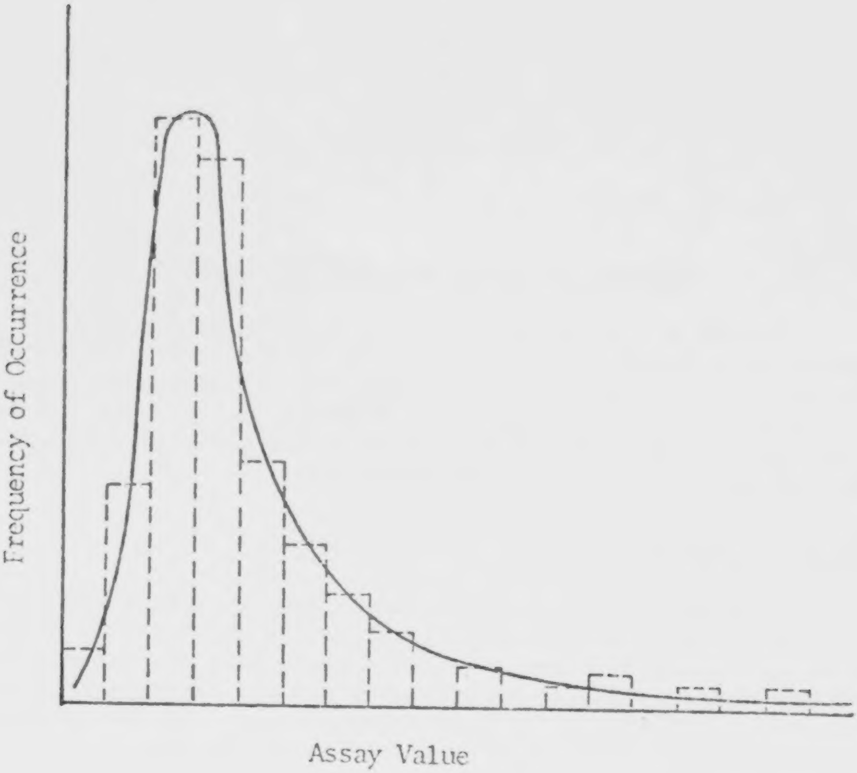
Figure 4. The lognormal distribution



Cut-off grade (% Cu)	Ore reserves (tons)	Metal content (tons)	Average grade (% Cu)
0.00	200,000,000	1,000,000	0.50
0.50	73,580,000	735,800	1.00
1.00	27,070,000	406,000	1.50
1.50	9,960,000	199,100	2.00
2.00	3,660,000	91,600	2.50
2.50	1,350,000	40,400	3.00
3.00	496,000	17,350	3.50
3.50	182,000	7,290	4.00
4.00	67,000	3,020	4.50

D. Economic criteria for mine development and operating decisions

Tonnage and grade estimates are made to assist economic evaluation of technically feasible alternatives for selection of the optimum economic alternative. The economic criteria which are used for evaluation purposes prescribe the types of ore reserve estimates required. Economic criteria for two types of decision problems are described here - criteria for the mine development investment decision and criteria for mine operating decisions.

Mine development decisions 5/

The mine development decision is made on completion of exploration to determine whether or not a mineral deposit should be developed to production and, if so, how. This involves the selection of an optimum mine development alternative from a number of technically feasible alternatives and the comparison of this optimum with those of other available investment opportunities. If the optimum is sufficiently attractive, the deposit is developed to production.

On the basis of available information and expertise, ore reserve, revenue and cost estimates are made for each technically feasible alternative. Economic evaluation techniques are applied to these estimates to assess the profitability characteristics of each alternative. These assessments provide support for the investment decision.

Ore reserve, revenue and cost estimates are based on limited information. Thus, profitability for decision purposes is more realistically expressed as "expected profitability". Associated with expected profitability is a degree of risk which reflects the reliability of information at the time of evaluation. To compare and select alternatives, both the expected value and the risk of

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5/ The analysis framework summarized here is based on a more detailed treatment of the subject (9).

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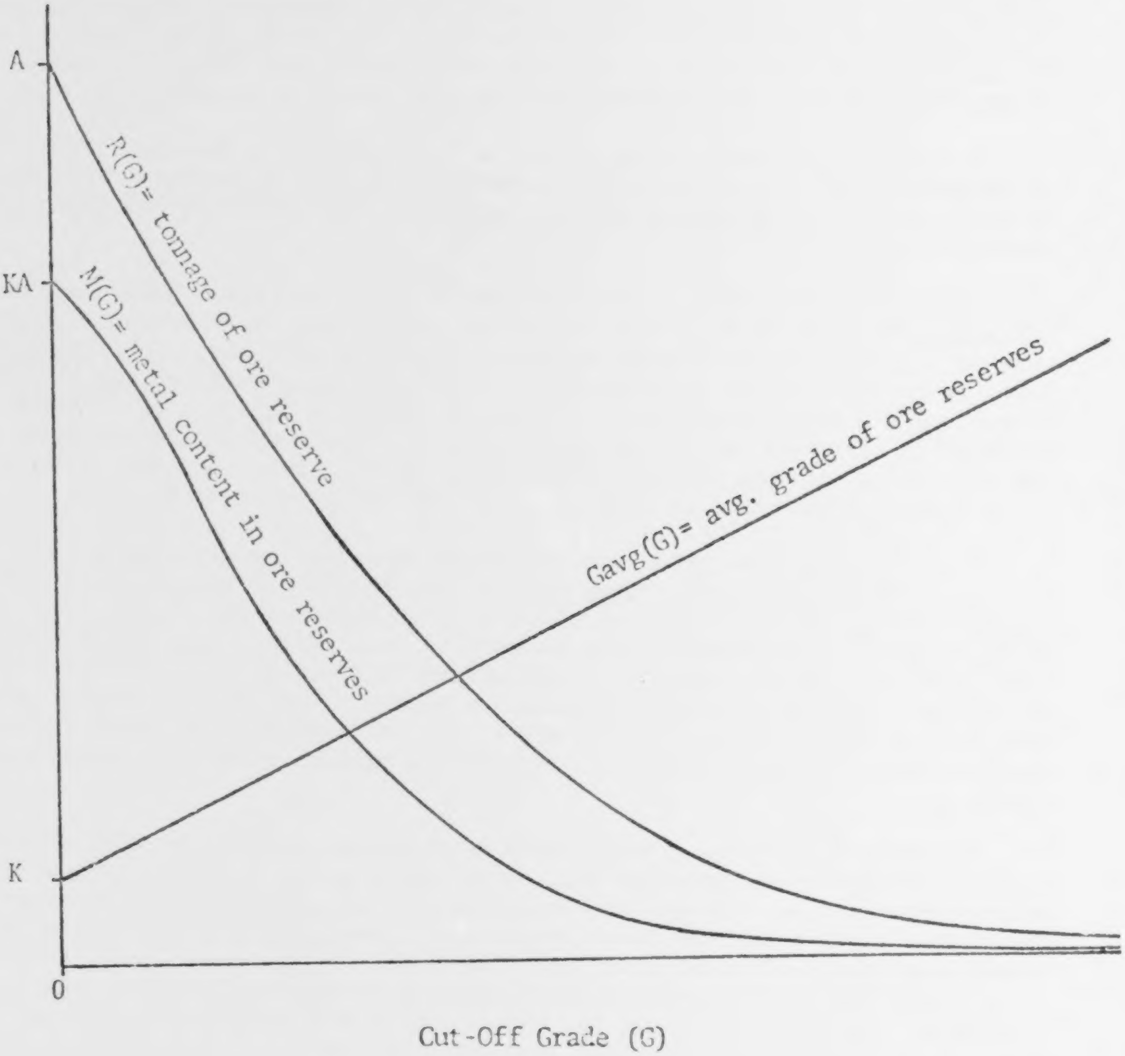
Figure 5. Exponential tonnage-grade relationship



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profitability should be evaluated. To do this, estimates of expected value and uncertainty are required for each of the input variables, including ore reserves. For this purpose, estimates of the mean (expected) value and the standard error of the mean are made for relevant tonnage and grade characteristics.

In addition to information which can be evaluated in terms of expected profitability and risk, intangible information must also be considered in the decision process. "Unknown economic resources" may constitute an important intangible.

Evaluation of expected profitability is based on single-point expected value estimates for each of the input variables. The Monte Carlo method simulates the risk of a profitability outcome by means of random sampling. This technique is based on the estimation of probability distributions to reflect the uncertainty of each of the input variables. The result of the simulation is an estimated probability distribution of the profitability outcome about its expected value. The critical component of this distribution is the lower confidence limit, reflecting the insurance of some minimum level of profitability.

Cash flow and time value are the basic concepts used to assess profitability. Net present value may be used to compare the profitability characteristics of alternatives, given a fixed level of investment (for example, fixed capacity). To estimate net present value, positive and negative annual cash flows are discounted at a predetermined interest rate representing the cost of capital, and the discounted values are summed. The rate of return, one of the most widely used profitability criteria, is the discount rate which equates the present value of negative cash flows with the present value of positive cash flows.

The selection of the optimum mine development alternative will depend on the economic criteria evaluated as well as on the size and risk preference of the mining organization. On the one hand, the mining organization, motivated by the desire for profitable investment, prefers a development alternative with a higher expected profitability to one with less. On the other hand, it is averse to risk and therefore prefers a development alternative with less risk to one with more. In economic terms, the organization will select the optimum alternative on the basis of maximizing its "utility".

Sensitivity analysis can be used to determine the relative importance of ore reserve input estimates to the profitability outcome. A porphyry copper example, developed by McCrea (10), is shown in figure 6. It can be seen that rate of return is very sensitive to possible changes in grade. In general, grade and metal price are the two most important input variables in evaluating the economics of mine development. Thus, estimation of the expected (mean) value and reliability of grade parameters is of critical importance to mine development investment decisions.

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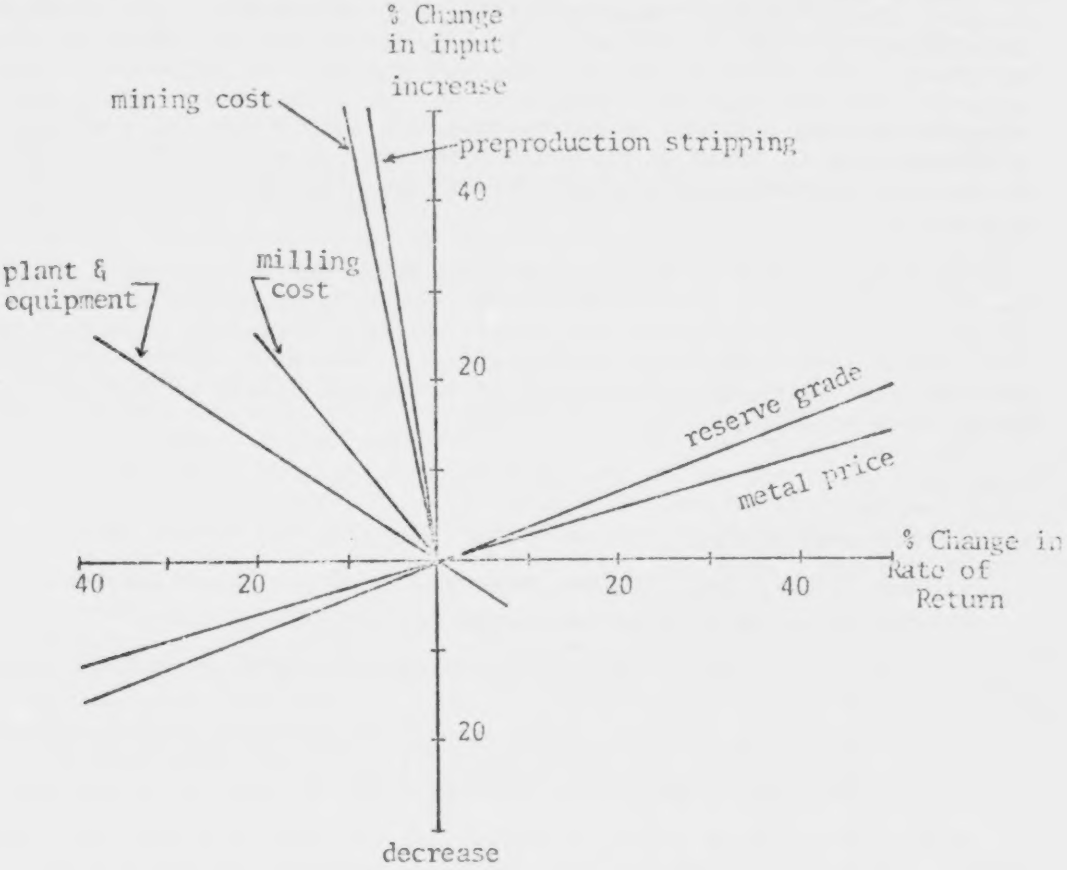
Figure 6. Porphyry-copper mine development; sensitivity of profitability to change in input variables



Expected Value Conditions

Capacity: 25,000 tpd
Avg. Grade: 0.4% Cu
Metal Price: $0.50 per pound
Rate of Return: 15.5%

Source: McCrea (10)

Mine operating decisions

For the mining operation, investment in exploration and mine development is sunk and decisions are of a tactical nature. Operational decisions embody such elements as cost control, production planning, productivity, the development and implementation of new mining and processing technology and equipment replacement decisions. The effectiveness of these policies will be reflected in cost trends which in turn will determine changes in cut-off grade, ore reserves and mine life. A primary objective of the mining organization with respect to a mining operation is maximization of total profit (cash flow) over mine life. This objective will be achieved by minimizing economic cut-off grade and maximizing ore reserves and mine life.

The cut-off grade is the minimum grade which can be economically mined. It is the "extensive margin of cultivation" discussed in classical economic theory. For operational conditions where the primary economic criterion is maximization of total profit over mine life, cut-off grade is determined by the equation of marginal cost and marginal revenue. At the margin, grade is just sufficient to offset cost, that is,

$$MC = MR = PGR/(1 + D)$$

where, MC = cost of mining and processing marginal ton of ore (in dollars);

MR = revenue realized from marginal ton of ore (in dollars);

P = net price of metal recovered (in dollars per ton);

G = cut-off grade, percentage content of mineral product in marginal ton of ore;

R = mill recovery factor (metal recovered/metal content of ore reserves);

D = dilution factor (waste mined/ore mined).

Thus, cut-off grade varies directly with the cost of producing a saleable mineral product and inversely with the price received for the product. As discussed in a preceding section, the significance of variations in cut-off grade for operational decisions depends on the geological parameters of the deposit, particularly the relationship which exists between tonnage and grade.

Operational decisions relating to ore reserve estimation fall into the category of "grade control". The time domain for this type of decision varies from the daily control required to minimize grade variability in mill feed, to production planning on quarterly, annual and five-year bases.

E. The economic sampling limit

The objective of mineral deposit sampling is to provide a data base for the tonnage and grade estimates required for the economic evaluation of alternatives.

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The costs of sampling are justified by the benefits which result from improving the reliability of the profitability criteria evaluated.

Tonnage and grade estimates are made to assist with mine development and mine operating decisions. A prior decision concerns the level of investment in sampling which is economically justified. This decision defines the economic sampling limit.

The reliability of tonnage and grade estimates is improved by increasing the sample size, thereby reducing the estimated standard error of the mean and the spread of confidence limits about the mean value estimate. However, the benefits from increased sampling are subject to diminishing returns, that is, as sample size is increased, the standard error of the mean is reduced but at a decreasing rate. At some point, marginal benefits from sampling investment just balance marginal costs. This point defines the economic sampling limit.

The economic benefit from increased sampling is the reduction of economic risk. A reduction in the standard error of the mean for ore reserve parameters reduces the standard deviation of the profitability outcome. This has the effect of increasing the lower limit of profitability. At the same time, costs are associated with increased sampling - the direct cost of the increased drilling or underground exploration required, as well as the time cost of delaying the mine development or mine operating decision. The time cost is the reduction in the present value of benefits which would result from the decision. These monetary and time costs reduce expected profitability and lower the entire profitability distribution. This has the effect of reducing the lower limit of profitability. Initially, benefits will exceed costs and the lower limit of profitability will be increased by further sampling. Economic risk will be minimized at the level of sampling where marginal benefits are equated with marginal costs. Sampling beyond this point will result in increased economic risk, that is, a reduction in the lower limit of profitability. Figure 7 illustrates the determination of the economic sampling limit based on the objective of minimizing economic risk. 6/

F. Ore reserve estimation for evaluating the economics of mine development

Ore reserve estimation for mine development decision purposes is normally based on the delineation of drilling results. Traditionally, the over-all estimation of size and grade parameters is made by analysing sample drilling data, using polygonal, triangular or cross-sectional methods. Using such geometric patterns, blocks within the deposit are given the average grade (or weighted average) of their samples. Choice of a particular geometric pattern is governed by the shape

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6/ This objective may not always be considered desirable. For example, in eastern European countries mineral deposits are fully explored before a development decision is taken. This sacrifices the economic benefits of more immediate development in the interests of encouraging the formulation of a more satisfactory long-term plan.

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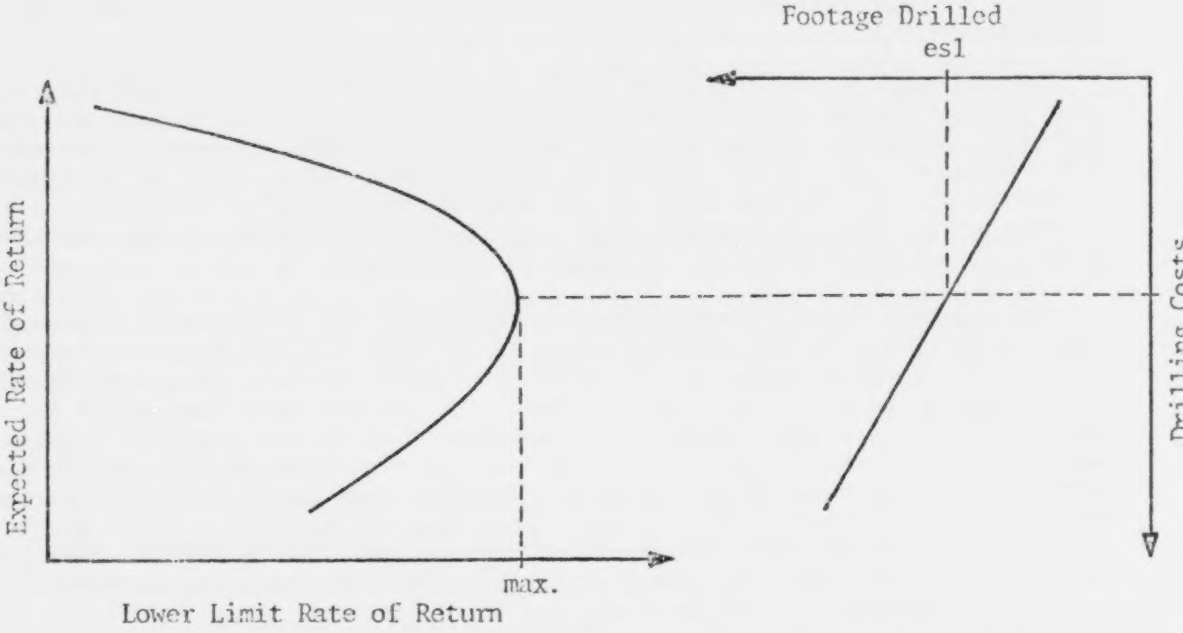
Figure 7. The economic sampling limit


Footage Drilled
esl
Drilling Costs
Expected Rate of Return
max.
Lower Limit Rate of Return

esl: economic sampling limit

and attitude of the deposit and the drilling pattern which has been used for delineation.

The polygonal method of computation assumes that the influence of a sample or the grade given by the sample extends halfway to the next sample. Use of the method involves construction of polygons about each of the drill-holes. The first step in the construction is to draw lines joining each drill-hole to every other adjacent drill-hole. This forms a series of triangles; and, if perpendiculars are erected from the mid-points of each of the sides of a triangle, they will meet at a common point. By joining these common points found for each of the triangles around a drill-hole, a polygon will be constructed about each drill-hole. The area of the polygon is computed and the thickness of the deposit within each hole is estimated. The average grade within each hole is assigned to the tonnage estimated on the basis of polygonal area and thickness. The tonnages of all polygons are summed and a weighted average grade computed to provide tonnage and mean grade estimates for the mineral deposit.

Computing size and grade by the triangular method involves constructing triangles with drill-holes at the corners. Lines drawn from each corner of a triangle to the mid-points of the opposite sides intersect at a point that becomes the common apex of three smaller equal-area triangles, each of which has one side of the original triangle as the base. The average grade of the point of intersection, which is assigned to the triangular area, may be computed by weighting the average grade of each hole inversely as the distance of each hole from the common point of intersection. An average thickness is computed in a similar manner. The tonnages associated with all triangles are summed, and a weighted average grade computed to provide tonnage and mean grade estimates for the mineral deposit.

Computation of size and grade using the cross-sectional method involves preparation of several sections of the deposit and plotting either the actual intersections or the projections of drill-holes on these sections. Computations of volumes and grades can be made in various ways. For example, computations of volumes and grades can be made in various ways. For example, computations of volume can be made by determining the average area of adjacent sections and multiplying by the distance between them. This volume is then assigned the average grade of the adjacent sections. The tonnages associated with all such segments are summed and a weighted average grade computed to provide tonnage and mean grade estimates for the mineral deposit.

The polygonal, triangular and cross-sectional methods used for estimating tonnage and mean grade for a mineral deposit assume that the area of influence of an assay in any one drill-hole is a function of the distance to any other adjacent drill-hole. In using these methods, no consideration is given to the mineralization which actually exists between the drill-holes. The methods are a function of geometry, which simplifies the calculations. They are not a function of the mineralization that the methods propose to predict.

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These traditional geometric methods have two major short-comings:

(a) Generally, the area of influence given to individual drill-hole assays far exceeds their actual area of influence. Drill-hole spacing is typically several hundred feet. Although the actual area of influence depends on mineral deposit conditions, statistical tests on a number of deposits have shown that the radius of influence of a sample grade may be as low as one foot and rarely exceeds 100 feet. Thus, the apparent precision of the geometric method in assigning drill-hole assays to large areas is fallacious. The actual mean grade contained within such an area may be quite unrelated to the drill-hole assays obtained. However, to the extent that results are used only to estimate over-all deposit parameters, this is not a problem in itself although the different weightings assigned to assays depending on relative distances between holes is incorrect and results in over-all bias;

(b) The geometric methods do not provide a measure of confidence in the over-all tonnage and mean grade estimates. A knowledge of the risk associated with these estimates, particularly grade, is important for evaluating the economics of mine development. Use of the geometric methods precludes risk analysis and the assessment of risk associated with profitability criteria. This limitation is not, however, inherent in the data. Rather, it occurs because these methods fail to fully utilize the data.

The application of statistical analysis does not have the above limitations. Using statistical analysis, individual sample values are not assigned to a particular part of the deposit. Rather, they are treated as random samples from the deposit as a whole and in this manner are used to provide over-all tonnage and grade estimates. Estimates are made for the mean value, standard deviation and standard error of the mean. This gives a basis for assessing not only the expected or mean values of the deposit parameters, but also the reliability or confidence of those estimates. Thus, there is a mechanism for evaluating the economics of controlling risk through additional sampling, as well as providing the type of probabilistic estimates required to assess the risk of mine development alternatives.

As outlined in a preceding section, the existence of trend within a deposit is non-random. However, if trend is known to exist, the deposit can be stratified into grade zones or other natural geological subdivisions to provide essentially random sets of data and permit the application of statistical analysis. This technique reduces error in estimating the expected grade of the total deposit being sampled and provides a more detailed basis for mine development planning. Some previous knowledge of the deposit is required to stratify the samples in this manner. Thus, it can usually only be used in the final stages of delineation sampling programmes.

The following example, concerning the Maggie Canyon manganese deposit in Arizona, was developed by Hazen (4) to compare tonnage and grade estimates by each of four methods - polygonal, triangular, cross-sectional and statistical analysis.

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Four separate exploration projects were carried out at Maggie Canyon between 1936 and 1950. In total, 42 surface drill-holes intersected the deposit, comprising 406 five-foot samples. The deposit was subdivided into three zones for estimating purposes - south, south-east and north. The results are shown in table 1.

Table 1. Tonnage and grade estimates: Maggie Canyon

Zone	Method of estimation			
	Polygonal	Triangular	Cross-sectional	Statistical analysis
South				
Size-mill (tons)	7.2	7.3	7.0	7.5
Grade (% Mn)	5.5	5.5	5.3	5.9
South-east				
Size-mill (tons)	10.6	10.5	11.0	10.1
Grade (% Mn)	4.2	4.4	4.0	4.1
North				
Size-mill (tons)	9.6	9.8	7.1	8.5
Grade (% Mn)	6.7	6.3	5.3	6.5
Total				
Size-mill (tons)	27.4	27.6	25.1	26.1
Grade (% Mn)	5.4	5.3	4.7	5.4

Results show close agreement for the grade estimates obtained using the polygonal, triangular and statistical analysis methods. The cross-sectional method results show the largest variation.

The application of statistical analysis to grade estimation for the total Maggie Canyon deposit is shown in table 2. The advantage of statistical analysis is that the reliability or confidence of the mean value estimate may be assessed. For example, using the Maggie Canyon case:

$N = 406$

$\overline{X} = 5.43\%$

$S = 4.41\%$

$S_{\overline{x}} = 4.41/\sqrt{406} = 0.22\%$

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Table 2. Statistical analysis: Maggie Canyon manganese deposit

(Sample frequency distribution and computations for 406 sample manganese assays from 42 surface drill-holes within deposit limits, assuming 2 per cent grade intervals)

Grade interval (GI)	Frequency (f)	Average grade weighted (percentages)	Midpoint of grade interval (mp)	Frequency times midpoint (f x mp)	Frequency times midpoint squared (f x (mp)^2)	Frequency times average grade
0.00- 1.99	69	1.39	1.00	69	69	95.91
2.00- 3.99	130	2.80	3.00	390	1 170	364.00
4.00- 5.99	79	4.84	5.00	395	1 975	382.36
6.00- 7.99	42	6.85	7.00	294	2 058	287.70
8.00- 9.99	35	8.95	9.00	315	2 835	313.25
10.00-11.99	15	10.77	11.00	165	1 815	161.55
12.00-13.99	19	12.84	13.00	247	3 211	243.96
14.00-15.99	5	14.71	15.00	75	1 125	73.55
16.00-17.99	4	16.59	17.00	68	1 156	66.36
18.00-19.99	1	19.96	19.00	19	361	19.96
20.00-21.99	3	20.88	21.00	63	1 323	62.64
22.00-23.99	2	23.05	23.00	46	1 058	46.10
24.00-25.99	0	0	25.00	0	0	0
26.00-27.99	1	26.35	27.00	27	729	26.35
30.00-31.99	1	30.70	31.00	31	961	30.70
	406	5.32		2 204	19 846	2 174.39

Mean grade estimate $= \overline{X} = \dfrac{\Sigma X}{N} = \dfrac{\Sigma f\ \text{(avg. grade)}}{N} = \dfrac{2{,}174.39}{406} = \underline{5.36\%}$

Approximate mean grade estimate $= \overline{X} = \dfrac{\Sigma(f)\ (mp)}{N} = \dfrac{2{,}204}{406} = \underline{5.43\%}$

Approximate standard deviation estimate $= S = \sqrt{\dfrac{\Sigma(f)\ (mp)^2}{N} - \left[\dfrac{\Sigma mp}{N}\right]^2} = \sqrt{\dfrac{19{,}846}{406} - \left[\dfrac{2{,}204}{406}\right]}$

$= \underline{4.41\%}$

Source: Hazen (4).

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If 95 per cent confidence limits are desired, a table of areas under the normal curve shows that these limits will lie ± 1.97 standard errors from the mean value estimate, that is, 5.43 ± (1.97) (0.22), or 5.0 per cent and 5.8 per cent. Thus, there is 90 per cent confidence that the mean grade of the total deposit will actually be between 5 and 5.8 per cent manganese.

Given size and grade estimates for the over-all deposit, ore reserves are derived for each alternative on the basis of assumed cut-off grade conditions. The cut-off grade associated with the optimum economic alternative will be used in practice. Since the economic evaluations generally involve complex interrelationships among variables and changing economic conditions over time, it is difficult to develop rules to guide initial cut-off grade assumptions. Nevertheless, it has been shown by Lane (9), Blackwell (1) and Taylor (13), that, generally, economic benefits result from sequencing mining so that cut-off grade is gradually reduced over mine life.

The optimization of cut-off grade is shown by Blackwell for a hypothetical open-pit copper mine development. The example assumes fixed mine and mill capacities and does not consider uncertainty. Thus, the net present value criterion may be utilized. The assumed mine development conditions are as follows:

Mine capacity: 40 million tons per year

Mill capacity: 15 million tons per year

Mill recovery: 85 per cent

Unit pit costs per ton mined: $0.25

Unit mill costs per ton ore: $0.50

Selling costs per ton copper: $50

Copper price: $0.39 per pound recovered

Tonnage and grade are specified by tonnage estimates over a range of grade categories. Economic results for assumed cut-off grade strategies are shown below:

Cut-off grade (% Cu)		Net present value (millions of dollars)	Total cash flow (millions of dollars)	Mine life (years)
Fixed:	0.2	255	910	25
	0.3	276	860	21
	0.4	294	819	18
	0.5	287	717	15
	0.6	236	548	15
Variable:	optimum	305	797	16

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From the results it can be seen that the optimum variable cut-off grade strategy has an advantage of $9 million over the highest present value obtainable by a fixed cut-off grade strategy. With this strategy, the cut-off grade is reduced from 0.54 per cent Cu in the first year to 0.17 per cent Cu at the end of mine life.

G. Ore reserve estimation for operational decisions

Ore reserve estimation for operational decisions must be much more detailed than for mine development decision purposes. These decisions involve tonnage and grade control issues. The planning horizon for these decisions varies from one day to several years. The following types of grade control problems may be considered:

(a) Daily grade control of mine production to minimize grade variability in mill fee, thereby optimizing mill efficiency and recovery;

(b) Cut-off grade decisions relating to closing of old working places and new stope development;

(c) Production planning, including cut-off grade considerations, on quarterly, annual and five-year bases;

(d) Control and economic limitation of dilution.

From the mill-operations point of view it is highly desirable that the grade of ore treated be as uniform as possible. Even with onstream analysis and computerized control, wide fluctuations are difficult to handle and result in loss of efficiency, reduced recovery and increased costs. Ore blending procedures in the mine should be directed towards minimizing these short-term variations.

Nevertheless, beyond some point, the cost of ore blending procedures becomes prohibitive. This means that mines cannot produce the same grade day after day. Grades will vary. However, over longer-term periods the average grade mined should be close to planned grade. This is required because of market commitments for concentrate sales and because of the needs imposed by financial planning for sources and uses of funds within the mining organization.

In some mining methods a certain amount of dilution is inevitable. In other cases it constitutes bad mining practice. Dilution should be classified into the following categories:

(a) Dilution with mineralized material means that the dilutant may carry recoverable mineral which may pay for its recovery;

(b) Dilution with material without mineral content is a more serious matter because this material will remove mineral from the ore in the recovery circuit.

For dilution control the greatest threats are hangingwall and footwall conditions. In many cases, the footwall is faulted, consisting of friable poorly mineralized rock which will be recovered in the mining cycle. Little can be done to eliminate dilution of this type. Dilution by hangingwall failure is time-dependent and structurally controlled. It can therefore be avoided if the danger is recognized. All other forms of dilution generally result from insufficient sampling information and bad mining practices. Dilution control requires a knowledge of the amount of dilution which can be economically accommodated and a flexibility in mining practice to adapt to unexpected changes in deposit conditions.

For purposes of grade control, tonnage and grade estimates are required for blocks associated with individual stopes and mining faces, and for stope-size blocks along development headings. The basis for these estimates is detailed stope and development samples, supplemented by mine car and skip sampling programmes.

The application of statistical analysis for tonnage and grade-control purposes is much more difficult than for the mine development situation previously outlined. For grade control, a small block must be assigned a grade based on a relatively large number of closely spaced samples. In this case, individual sample values are not random and cannot be assumed independent of each other. Thus, the methods of statistical analysis previously discussed are not applicable.

For statistical analysis of tonnage and grade in operating situations, a number of geostatistical techniques, primarily developed in France and South Africa, may find practical application. The concepts of regionalized variables, extension variance and the variogram, as described by David (3), may be applied to determine optimal weights to assign to individual samples to obtain the best estimator for valuation of blocks. Krige (6) has applied correlation and regression analysis to the selective mining of gold ores in South Africa. A regression equation was employed to estimate the unknown face value as a function of the face values of approaching stope panel faces. These techniques provide the statistical bases for predicting grade contour surfaces for grade control purposes.

Given tonnage and grade estimates for individual working blocks within the deposit, economic conditions and criteria must be applied for decision purposes. Economic conditions include operating costs, dilution and recovery factors, market demand, price and government policy considerations. Estimates of cost, recovery and dilution conditions are required on individual stope and individual working face bases for meaningful control. Thus, a detailed mine costing system structured for decision rather than accounting purposes is a prerequisite.

The economic criteria applied for tonnage and grade control decisions are relatively simple. Generally, marginal analysis may be used. Since investment in mine capacity is sunk, time value and profitability are less important than in the mine development decision situation. Maximization of total profit is a primary objective. This will be achieved by mining to the economic margin. However, over the longer-term planning horizon, economic benefits may be realized from sequencing operations so that cut-off grade is reduced over time. The desirability of mining to the true economic margin will increase as the deposit becomes depleted.

The determination of cut-off grade and ore reserves for a mining operation may be strongly influenced by government policy considerations. Mineral conservation policy may require mining to the average grade of ore reserves as defined by the true economic margin. The level and structure of mining royalties and taxation may act to raise cut-off grade. These effects are illustrated by a hypothetical example.

A copper deposit (based on an earlier example) is assumed containing 200 million tons at an average grade of 0.5 per cent copper. The tonnage-grade relationship is exponential. A mine capacity of 3 million tons of ore per year has been installed at a capital cost of $37.5 million. Other operating parameters are as follows:

Operating cost: $3.60 per ton of ore mined
Recovery factor: 80 per cent
Dilution: nil
Price: $750 per ton of copper recovered in concentrate

Three levels of mining taxation (T) are considered over the mine's productive life: $50 million, $70 million and $90 million.

The example compares the optimization of the mining operation at each level of taxation for two forms of taxation:

Profit tax: t_p, percentage of taxable profit; operating and capital costs are the only allowable deductions

Value of production tax: t_{vp}, percentage of revenue

It is assumed that the objective of the mining organization is to maximize total profit over mine life. Time value is neglected in the interest of simplicity.

Consider the $50 million level of taxation.

(1) Profit tax

$$G = \frac{MC}{PR} = \frac{3.60}{750(.8)} = 0.60\% \text{ copper}$$

- applying the exponential tonnage-grade distribution as described above with A = 200 million tons, K = 0.5%, G = 0.6%:

Ore reserves = 60,239,000 tons
Copper content = 662,600 tons
Average grade of ore reserves = 1.10% copper

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Total revenue	= 662,600 (.8)(750) =	$397,578,000
Operating cost	= 60,239,000 (3.60) =	216,860,000
Operating profit		180,718,000
Capital cost		37,500,000
Taxable profit		143,218,000
Mining tax (T)		50,000,000
Total profit (cash flow)		93,218,000

$$\text{Tax rate} = t_p = \frac{50{,}000{,}000}{143{,}218{,}000} = 34.9\%$$

$$\text{Mine life} = \frac{60{,}239{,}000}{3{,}000{,}000} = 20.1 \text{ years}$$

(2) Value of production tax

$$G = \frac{MC + t_{vp}(PRG)}{PR} = \frac{.006}{(1 - t_{vp})} \tag{1}$$

$$T = t_{vp} \text{ (copper content of ore reserves) } (PR) = \$50{,}000{,}000$$

$$\text{Copper content of ore reserves} = \frac{50{,}000{,}000}{600\ t_{vp}} \tag{2}$$

Solutions for G, t_{vp} and copper content of ore reserves are obtained by solving expressions (1) and (2) simultaneously with the exponential tonnage-grade relationship:

Cut-off grade = 0.70% copper
Ore reserves = 49,319,000 tons
Copper content = 591,800 tons
Average grade of ore reserves = 1.20% copper

Total revenue	= 591,800 (.8)(750) =	$355,100,000
Mining tax (T)		50,000,000
Operating cost	= 49,319,000 (3.60) =	177,550,000
Capital cost		37,500,000
Total profit (cash flow)		90,050,000

$$\text{tax rate} = t_{vp} = \frac{50{,}000{,}000}{355{,}100{,}000} = 14.2\%$$

$$\text{mine life} = \frac{49{,}319{,}000}{3{,}000{,}000} = 16.4 \text{ years}$$

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The complete set of results for this example is set forth below:

Parameter	T = \$50 million		T = \$70 million		T = \$90 million	
	t_p	t_{vp}	t_p	t_{vp}	t_p	t_{vp}
Cut-off grade (% Cu)	.60	.70	.60	.765	.60	.865
Ore reserves (million tons)	60.2	49.3	60.2	43.5	60.2	35.8
Copper content (thousand tons)	663	592	663	548	663	485
Total revenue (million \$)	397.6	335.1	397.6	328.8	397.6	291.0
Operating cost (million \$)	216.9	177.5	216.9	156.6	216.9	128.9
Capital cost (million \$)	37.5	37.5	37.5	37.5	37.5	37.5
Total profit (million \$)	93.2	90.0	73.2	64.7	53.2	34.6
Tax rate (percentages)	34.9	14.2	34.9	21.3	34.9	30.9
Mine life (years)	20.1	16.4	20.1	14.5	20.1	11.9

The example illustrates the differing effects of profit and value of production forms of mining taxation on cut-off grade and ore reserves for three levels of taxation. Obviously, such government policy considerations have an important impact on operational cut-off grade decisions.

H. Summary

The paper reviews statistical and economic principles relating to the estimation of tonnage and grade for mineral deposits and the assessment of ore reserves.

The definition of ore reserves for estimating purposes should embody distinct geological and economic dimensions. Geological estimates of tonnage and grade for the known part of a mineral deposit establish a base to assess ore reserves given various sets of economic conditions.

A knowledge of the geological environment is essential for the specification of grade trends within a deposit and for understanding the relative importance of tonnage and grade parameters and the significance of variations in cut-off grade.

Statistical analysis of sample data provides expected value estimates and confidence limits for tonnage and grade parameters. These types of estimates are required to evaluate expected profitability and risk for mine development alternatives. Economic criteria for mine operating decisions are based on marginal analysis.

Sampling improves the reliability of estimates for economic criteria. However, the benefits from increased sampling are subject to diminishing returns and, at some point, the marginal benefits from improved reliability are just sufficient to balance the marginal monetary and time costs incurred. This defines the economic sampling limit.

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Traditional geometric techniques for tonnage and grade estimation do not provide a measure of confidence in the mean value estimates. While statistical analysis does not have this limitation, problems associated with non-random data and grade trends have to be overcome for realistic application. The optimization of cut-off grade for mine development is usually achieved by sequencing mining so that cut-off grade is gradually reduced over mine life.

Detailed tonnage and grade estimates based on stope and development sampling programmes are required for tonnage and grade control at operating mines. Geostatistical techniques utilizing correlation and regression analysis may be usefully applied for these purposes. Detailed economic conditions are imposed on the tonnage and grade estimates obtained to define ore reserves for economic analysis of mine planning and operational control alternatives. Government mineral policies, particularly those relating to royalty payments and taxation, may have important effects on operational cut-off grade and ore reserve estimates.

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II. EXPLOSIVES: PRODUCTS AND BLASTING IN CANADIAN MINES

W. B. Morrey*

The first commercial use of explosives in Canada is unknown. It probably involved black powder in some form of agricultural, construction or mining venture typical of the frontier days of this nation. The first known commercial factory was built in 1862 to manufacture black powder.

In 1972 the total production of commercial explosives in Canada was close to £300 million. During the past century there have been many major developments responsible for the impressive expansion of this industry:

(a) The construction of the transcontinental railways in the late 1800s;

(b) The emergence of the base-metal industries in Ontario, Manitoba and British Columbia in the early 1900s;

(c) The "post depression" expansion of gold mining commencing in the early 1930s;

(d) The development of the iron-ore resources of Quebec and Labrador in the 1950s;

(e) The construction boom of the late 1960s, principally in the form of hydroelectric projects and massive highway and railroad extensions;

(f) More recently, the opening of the large copper-molybdenum mines in British Columbia and the massive revival of the coal industry in several areas of western Canada.

Canada is a large country. The land mass is slightly less than 3 million square miles. This figure has a specific influence on the design and use of explosives. Within these boundaries can be found almost every known type of sedimentary, igneous and metamorphic rock. Rock and mineral hardness ranges from talc to taconite. Temperature is a major factor. Explosives must store, handle and perform in temperatures ranging from -60°F to +100°F. This vast geographical area, with its mining centres scattered in many frontier areas, has stimulated the "satellite" or "on-site" explosives manufacturing and delivery system. Even the most remote property can benefit from large-volume production at the lowest possible cost delivered into the borehole with full assurance of quality control and performance of even the most intricate of modern technological developments.

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A. Classification of explosives

All Canadian explosives fall into two basic categories according to their initiation sensitivity which is a measure of their potential hazard. Any material sensitive to initiation by a single No. 8 strength detonator is classed as a "high explosive". Material that is not sensitive to a single No. 8 strength detonator is classed as a "blasting agent" and enjoys lower transportation rates, higher capacity of storage magazines and, in most cases, freedom for bulk blending or on-site manufacture according to the federal regulations.

Explosives

High explosives	Blasting agents
Sensitive to a No. 8 cap	Not sensitive to a No. 8 cap
Nitroglycerine dynamites	AN/FO mixtures
Cast primers (pentolite)	Metallized AN/FO mixtures
Cap sensitive slurries	Packaged dry blasting agents
	Slurry blasting agents TNT or NCN types

Cap-sensitive slurry explosives represent the newest technical development in the explosives industry. These will be covered in detail later in this paper.

High explosives

Dynamites - collectively, all nitroglycerine-sensitized, cartridged explosives - are now rarely used in primary blasting in open-pit mines. They have been replaced by slurries and ammonium nitrate/fuel oil (AN/FO) mixtures which provide much higher efficiency and a drastic improvement in safety.

Dynamite still plays an important role in underground mines, particularly where wet conditions exist such as in shaft sinking, longhole blasting, wet raises and development headings.

The basic types are shown in the following table. These encompass a strength range of 20 per cent to 95 per cent. This classification is based primarily on water resistance and product density. Generally, the cost decreases in line with decreases in these two basic properties.

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Type	Major properties
Gelatin dynamite	Excellent plasticity, water and pressure resistance, high strength
Ammonia gelatin dynamite	Good plasticity and water resistance, broad range of densities and strengths
Semi-gelatin dynamite	Medium strength and density, fair plasticity and water resistance
Ammonia dynamite	Low-density, non-plastic, low-water resistance
Permitted dynamite	Special formulations for underground coal mines

These products are available in a wide variety of cartridge sizes and types ranging from 7/8 x 8" to a maximum of 8" diameter x 50 pounds. Cartridges are of convolute paper wrap or spiral wound with tapered crimp ends to facilitate loading in rough holes. A special cartridge, 5/8" diameter x 24" long, with sleeve couplers is used in the perimeter blasting and pre-shearing techniques for stope backs, development backs and pillar walls.

1. AN/FO mixtures

Ammonium nitrate - fuel oil mixtures were introduced into Canadian surface mining operations in 1957 and into underground mines beginning in 1960. Acceptance was immediate, and these low-cost, high-performance mixtures, enhanced by their inherent safety features, soon established themselves in large volume in the mining industry. The introduction was accelerated greatly by the availability of high quality porous prills of ammonium nitrate developed earlier for the fertilizer industry. Improvements in prill quality and anti-caking additives quickly followed. Special "blasting grade" prills were developed that had good storage and flow properties together with excellent oil absorption. Intimate and permanent blending of the solid and liquid components assured good sensitivity and high performance.

Small volume users blend on the bench by simply pouring the required amount of oil into an open bag of prills. Although simple, this is, in fact, explosives manufacturing. Special permission is required from the Government through the Office of the Chief Inspector of Explosives. He will authorize such blending only for immediate use. Field-blended mixtures cannot be stored - they must be used immediately after mixing.

Portable AN/FO mixers

Where boreholes in the 4-7" range are used, blending and loading by single bags does not provide the full economy potential of this product. A lightweight unit designed to produce an accurately blended AN/FO product at a rate consistent with requirements of small- to medium-sized blasting operations, was developed. This unit can be mounted on any truck and operated hydraulically by means of a power take-off from the truck transmission.

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This portable AN/FO mixer consists of an aluminium hopper of 300 pounds capacity and a tubular 6" diameter screw auger attached to a sturdy frame. A hydraulic motor, operated by the power take-off, drives the auger and a positive displacement fuel-oil pump feeds oil into the auger through a manifold system. The fuel oil pump and auger are coupled to the hydraulic motor by means of a chain and sprockets, giving a direct mechanical drive which ensures a balanced 94 per cent AN-6 per cent oil mixture irrespective of the delivery rate.

Loading rates can be varied from 150-300 pounds per minute at power take-off speeds of 1,000-2,000 rpm. Feed is controlled by the operator at the discharge end of the auger and is measured by a calibrated revolution counter attached to the auger.

Bulk AN/FO delivery

Large-scale users of AN/FO have hole diameters of 9 7/8" to 12 1/4". Loads of 28 and 42 pounds/foot of borehole require high capacity units and rapid loading rates.

Two types of bulk AN/FO mix trucks are used. One type uses an aluminium box-type body for carrying ammonium nitrate prills at payloads of 10,000-30,000 pounds. The AN/FO mixing and conveying unit is mounted externally at the rear of the unit. This truck is equipped with a power take-off, hydraulic hoist mechanism and all necessary controls and motors for loading operations. The truck may be filled from a storage silo or by auger feed directly from rail hopper cars. During delivery the box is raised so that the AN prills are fed by gravity to a transverse screw-conveyor. This maintains a constant head of prills over the feed unit.

A pneumatic system can be used where low pressure air (15 psi) conveys the prills from the feed unit through the delivery hose. Fuel oil is metered into the prills - air stream as it leaves the feed unit and mixing takes place through the turbulence as the AN/FO passes along the delivery hose. Feed can be controlled by single valves on either side of the unit or by a remote control attachment that allows the blaster complete control at the borehole. Loading rates are approximately 300-700 pounds per minute. This unit has a low centre of gravity and can operate over very rough ground. It can also operate as a service truck when not in use for AN/FO loading. The loading hose is generally three inches in diameter and fifty feet long, permitting several holes to be loaded from each stop.

The second type of unit involves straight auger feed. The hopper body is V-shaped, and the prills are fed by gravity into a second auger operating as a swinging boom. Oil is metered into this second auger near the feed end. Discharge rates vary from 250 to 500 pounds per minute.

The energy of AN/FO mixtures can be improved substantially by the addition of powdered aluminium to an oxygen balanced blend. The following table shows the increases in relative weight-strength and bulk-strength as the metal content increases. These values are based on a standard AN/FO mixer having an established value of 100.

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Metal content (%)	Density (gm/cc)	Relative weight-strength	Relative bulk-strength
0	0.83	100	100
2.5	0.85	110	110
5.0	0.86	118	120
7.5	0.87	125	127
10.0	0.88	133	138
12.5	0.89	139	147
15.0	0.90	146	155

Bulk metallized AN/FO mix trucks were introduced in 1972. The first unit to operate in North America is illustrated here. The prill storage hopper is affixed to the rear of the prill bin and holds 3,200 pounds. The unit can add aluminium to AN/FO mixtures in a range of 1-18 per cent. This unit can also provide straight AN/FO mixtures with no aluminium. Varying percentages of metal may be added for bottom and intermediate loads, topped off with a non-metal mixture.

A rotary feeder for the aluminium is fixed to the bottom of the aluminium bin and feeds a measured quantity of aluminium into the atmospheric side of the main stainless steel air lock. The quantity of aluminium is governed by the revolutions of the rotary feeder. Two hydraulic valves shown below the oil-flow meter control the rotary aluminium feeder, and the main hydraulic drive system that motivates the main longitudinal AN feed auger and oil delivery.

The oiled AN-aluminium blend is pneumatically blown into the borehole by a low-pressure air system.

As aluminium can constitute a fire hazard, many safety features have been incorporated, including a quick-dump hatch in the aluminium bin in the event of fire within the bin.

Borehole dewatering

Field applications confirmed quickly that metallized AN/FO mixtures had tremendous potential in open-pit mines. Even in hard magnetite iron ore, they could replace metallized slurries of much higher density in bottom as well as intermediate and top loads. The only deterrent was the presence of water in most open-pit mines where these high-energy levels would apply as bottom loads. Borehole dewatering has been practised in Canada for at least 15 years with varying success. Holes were blown out with compressed air or pumped out with submersible air or electric pumps. Blowing by compressed air was not effective owing to serious depth limitations and incomplete removal.

Submersible pumps seemed subject to frequent breakdown. Electric pumps are considered a possible hazard in conductive, metallic sludge.

Early in 1972 a new design of hydraulic pump was introduced that has performed with complete success under all field and climatic conditions.

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A typical unit consists of two "down-the-hole" pump elements mounted on a four-wheel drive truck with power take-off to drive the two pumps. The 7 stage pump measures 3 5/8" diameter by 22" long. This pump unit is enclosed in a protective rubber sleeve, giving total outside dimensions of 4 3/4" by 42". For efficient operation, minimum wear and to avoid jamming the pump in the borehole, it should not be used in holes less than 6 1/2" in diameter.

The hose reels are also hydraulically driven. They normally hold 75 feet of hose assembly consisting of an outer protective hose encasing one hydraulic down-line, two hydraulic oil return lines and one 1" diameter water discharge hose. A special 125 foot hose assembly has been used to dewater successfully holes 110 feet deep.

An extendible boom and sheave wheel is part of each hose reel. The maximum span, with both booms fully extended, is 21 feet. The truck can be driven between rows of holes to dewater two holes simultaneously and independently at a rate of 40-60 gallons per minute.

The pumped water flows through a 2" discharge hose connected at the rear hub of the hose reel. Usually, this discharge hose is strung along the bench or over the brow. In summer it can be discharged into a tank truck for wetting mine haulage roads.

In freezing weather, on completion of dewatering, the system is flushed with a coloured methyl alcohol or glycol solution to prevent damage to the pump and water hoses. The anti-freeze solution is reclaimed before beginning the next dewatering cycle.

Plastic borehole liners are used in all dewatered holes. The basic requirement in the use of liners is that the hole be completely empty of water when the AN/FO is loaded into the liner. These mixtures are not heavy enough to sink and completely expand the liner against hydrostatic pressure. Considerable speed is required to place and fill the liner before water re-enters the hole.

Packaged dry-blasting agents

Although the techniques of dewatering and borehole lining have been perfected to a high degree, there are locations where water problems cannot be overcome economically by these means. In most cases, AN/FO can do the job if the product density is increased and a suitable water-resistant package is used. These products are referred to as "dry" blasting agents in comparison to slurry blasting agents which may employ up to 15 per cent water as a basic ingredient.

These products are based on high density grained ammonium nitrate with a liquid organic sensitizer. They have a minimum density of 1.1, and thus will sink in water. Water resistance is provided by a double polythene bag, the inner heat-sealed, the outer wire-tied. These bags are within a rigid, spiral-wound tube container with a tapered plastic end cap. These outer tubes protect the plastic bags during transportation, handling and loading. Their rigid nature also ensures a uniform column load. Water resistance is unlimited, provided the polythene bags are not damaged. Careful handling is required.

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A straight sensitized AN mixture is used for top loads or in easy breaking formations. Three metallized grades are available for harder formations and as toe loads. These products are available in diameters of 4-8 1/2" with the following basic properties. Strength values are relative to a standard AN/FO mixture equal to 100 units.

Metal content (%)	Velocity of detonation (ft/sec)	Density (gm/cc)	Relative weight strength	Relative bulk strength
0	11,500	1.11	100	137
5	11,500	1.11	119	156
10	11,500	1.11	133	175
15	11,500	1.16	147	203

2. Slurry blasting agents

The world's first commercial slurry explosives were developed in Canada and introduced to the Canadian mining industry in 1958. This basic concept is now used throughout the world to provide bulk and packaged products of high energy and water resistance, good coupling properties plus an extreme level of manufacturing and performance safety.

The first slurry explosives were characterized by the use of "self-explosive" sensitizers. These are sensitizers that are themselves explosives but not necessarily sensitive to initiation by a single No. 8 strength cap. TNT is the most common sensitizer in this class. It is still widely used where reliability, high brisance, excellent water resistance, low temperature performance and immunity to the desensitizing effect of column or hydrostatic pressure are required. Other self-explosive sensitizers are smokeless powder, nitrostarch and a variety of organic nitrates.

A second family of slurry explosives is characterized by the use of "non-self-explosive" sensitizers. These may be best described as "sensitizing mechanisms" which usually involve the use of micro-bubbles of air or other gas. These micro-bubbles may be used alone or in conjunction with organic nitrates or with metallic aluminium of very fine particle size. Micro-bubbles, through adiabatic compression and the energy of impingement of metallic particles against the inner bubble wall create the multiple "hot spots" that develop and sustain the detonation reaction throughout the column. None of the ingredients is explosive - only the finished products.

These products have become known as NCN (nitro-carbonitrate) slurries based on terminology borrowed from early American transportation classification. In comparison with TNT-based slurries, these products have: lower density; lower velocity and energy; less resistance to water; less resistance to pressure; and inferior low-temperature performance.

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They find their greatest use in medium-hard rock where conditions of exposure, pressure and temperature are not severe.

I wish to emphasize that the slurry products mentioned so far have one important and critical property in common. They are not sensitive to initiation by a single No. 8 strength detonator and thus fall into the "blasting agent" category. This low sensitivity gives excellent safety features but results in critical diameters that preclude their use in holes less than three inches in diameter. They require high-energy boosters to ensure positive initiation under all field conditions.

Both TNT and NCN slurries are available in a wide variety of strengths to suit the broad range of rock and ores encountered. The basic properties are shown in the table.

Table of properties: TNT and NCN slurries

Product	Average density (gm/cc)	Velocity of detonation (fps)	Relative strength	
			Bulk	Weight
TNT-sensitized slurries				
HYDROMEX	1.55	18,000	166	90
HYDROMEX T-1	1.32	13,800	126	80
HYDROMEX T-3	1.50	18,000	159	89
HYDROMEX T-9	1.55	19,300	162	90
HYDROMEX LD	1.25	16,500	125	84
HYDROMEX T-3 LD	1.25	16,500	126	85
HYDROMEX M-210	1.54	16,200	212	117
HYDROMEX M-313	1.54	16,200	226	125
HYDROMEX M-415	1.54	16,200	234	130
HYDROMEX M-620	1.57	15,500	259	141
HYDROMEX M-825	1.60	15,000	280	153
NCN slurries				
NITREX 201	1.23	13,000	95	65
NITREX 304	1.26	13,000	126	84
NITREX 407	1.29	13,000	146	95
NITREX 510	1.31	13,000	166	107
NITREX 613	1.33	13,000	183	115
NITREX 716	1.34	13,000	195	123
NITREX 819	1.36	13,000	209	129

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Bulk slurry loading

There are two basic systems of bulk loading slurry explosives - by mix truck and by pump truck. The slurry mix truck - the first type developed - is a truck-mounted continuous-mix process plant for producing slurry and pumping it directly into boreholes through a loading hose. Mix trucks carry a liquor of ammonium nitrate, sodium nitrate and, possibly, other oxidizing salts in a tank maintained at 110-130°F.

Solid fuels and gelling agents, separately or pre-mixed, are carried in hoppers. The liquor from the solution tank and the dry ingredients from the hoppers are fed in measured quantities into a mixing chamber, then to a positive displacement pump and through 50-100 feet of 1 1/2-2" loading hose into the borehole. The action of the pump supplements the mixing that takes place in the mixing chamber to produce a homogeneous product. The mixed ingredients are in a viscous state by the time the slurry is discharged into the borehole. Special cross-linking agents create a stiff, waterproof gel in minutes.

Transfer of powdery or greasy materials from hoppers is commonly done by auger feeders. Granular materials such as TNT or aluminium are fed by belts.

Normal production capacities of mix trucks are 15,000 to 25,000 lb per trip; the capacity of the liquor tank is usually the limiting factor. Pumping rates are normally around 400 lb per minute but lower or higher rates are easily attainable. All NCN-type slurries are manufactured and pumped from mix trucks.

The slurry pump truck is more commonly used for TNT-sensitized slurries. This type of system differs from the slurry mix truck in that the slurries are manufactured in a satellite plant remote from the blasting operations rather than in the truck itself. The slurry pump truck is basically a bulk carrier of semi-thickened TNT slurry which may be formulated so that no additives are required when it is pumped into the hole; or the slurry may require the addition of extra thickener, gelling agent or both at the time of pumping. Minor ingredients are commonly added in solution or in suspension in glycol by a small separate pump discharging into the hopper intake of the slurry pump. The Moyno, advancing-cavity type pump is used.

A typical pump truck employs two horizontal, stainless steel tanks mounted on a tilting frame. This permits a low centre of gravity while moving over rough terrain, a constantly positive head of slurry over the pump hopper and complete emptying of the tanks. These trucks may carry 10,000 to 40,000 lb per trip. Different grades may be carried in each of the two tanks. Pumping rates may be varied from 300-650 lb per minute.

3. Satellite plants

The vast size of Canada, the widely scattered locations of the major mining companies, the high transportation rates for all explosives finished goods and the high volumes required, all contribute to the evolution of the satellite plant concept for bulk slurry and AN/FO products. As these mixing plants supplement production from large, longer-established plants, the term "satellite" has been

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coined. They are located as close as possible to the mining operations. Bulk trucks are designed primarily for mixing and loading on the bench - not for extensive highway travel.

A typical operation consists of: materials handling equipment for unloading rail-cars or other bulk transport; facilities for storage of raw materials; mixers for blending dry ingredients; heated process facility for preparing and storing non-explosive solutions; and custom designed trucks to deliver bulk explosives directly into the borehole.

Plants servicing TNT pump truck operations include heated holding tanks where up to 20,000 lb of pre-thickened slurry are held awaiting transfer to the pump truck. These are refilled while the truck is operating on the bench.

4. Primers and priming techniques in open pits

All blasting agents have low initiation sensitivity. AN/FO sensitivity varies with the type of prill, the oil content and the moisture content. Even with borehole liners there is no positive guarantee that water will not penetrate through punctures and abrasions. The sensitivity of TNT slurries decreases as the TNT content is reduced. As a class, NCN slurries have the lowest initiation sensitivity. All slurries become less sensitive at low temperatures and the severity of Canadian winters makes this a significant property.

A high level of priming is a basic requisite. The primer must develop high energy and, particularly, high detonation pressure. It must have good receptor sensitivity for use with detonating cord; it must be physically and chemically stable to withstand exposure to salt, saline solutions and hot liquors.

The cast pentolite primer was the first type developed. These are cylindrical primers using a 50 per cent TNT, 50 per cent PETN composition. This primer has two longitudinal tunnels for detonating cord attachment.

As lower sensitivity slurries were developed, particularly those with higher metal contents, the need for larger and stronger primers developed. The 1 lb primer is the general standard today but primers of 1 1/2 lb, 4 lb and 5 lb are available. Considerable research has been done to determine the optimum priming of AN/FO. It has been concluded that the optimum primer needs only two basic properties: a high detonation pressure; and a diameter equal to that of the borehole.

This can be easily and economically achieved by embedding a 1-lb pentolite primer in a bag of TNT slurry.

At least two primers should be used in every blast hole - it is false economy to skimp on primer size or number.

The bottom primer which is generally positioned in a slurry toe load, should be placed inside a bag of slurry, even though the hole is loaded with bulk explosive. This protects it and centres it properly in the hole. With package loading, this primer bag should be at the bottom of the hole, particularly during winter operations when the product is stiff. If the primer were positioned above

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the bottom, slumping of the charge through thawing and the pressure of overlying cartridges could cause column separation away from the primer and also subject the detonating cord downline to serious tensile strain.

When bulk loading, a packaged primer is preferred at the bottom of the hole so that pumping can be continuous from the bottom without interruption. If only a cast primer is used, it should be located about five feet above the bottom of the hole. There is considerable turbulence during initial pumping and this action on water and sludge can cause contamination in the critical primer location if it is too close to bottom.

B. Initiating devices and methods

Safety fuse and "Thermalite Ignitercord"

Safety fuse is used in most underground mines in development headings, raises and in many forms of small-hole stoping. The nominal burning time is 40 sec/foot ± 10%. This fuse is specially constructed to withstand storage and handling in the cold Canadian climate. It has a special interior plastic sheath over the black powder core to prevent ingress of mixture or diesel oil from AN/FO mixtures.

"Thermalite Ignitercord" is widely, if not exclusively, used as a fuse igniter in all underground primary blasts. This is because of the long-established safety features, the reliability of components and the assured positive rotation of all delayed blasts. Special Ignitercord connectors are crimped to the fuse ends to provide a water resistant closure and positive seating against the fuse end. The Ignitercord fits into the slotted end and is firmly secured by pressing the metal flap shut. This product is available in packages of 33 1/3 ft and 100 ft with three specific burning speeds:

Fast : 3-5 sec/foot

Medium : 5-10 sec/foot

Slow : 15-21 sec/foot

Blasting caps are available in No. 6 strength for general use with high explosives and in high strength for initiating pneumatically loaded AN/FO without any additional primers or boosters.

Over the past few years, the mining industry has converted almost entirely to blasting cap-safety fuse assemblies. These are pre-cut, factory assembled units of safety fuse with Ignitercord connectors and blasting caps attached. They are available in the following standard lengths which are colour coded for identification (in feet): 7, red; 8, orange; 9, purple; 10, green; 11, brown; 12, yellow; 13, white; and 14, blue. These are individually packaged in bundles of 10, with 200 units in each shipping container.

These factory-made assemblies relieve the mines of the labour and equipment costs of assembling components. They also provide higher quality of assembly and

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control. A major safety feature is the incorporation of an electrical shunt inside the blasting cap to ensure that this cap cannot be prematurely initiated by discharge of static electricity during handling in cold weather or during pneumatic loading of AN/FO mixtures. A metal staple is driven into the fuse so that the legs of the staple penetrate into the black powder core. The crown of the staple is exposed on the surface of the fuse. The blasting cap is crimped in place with a normal two-ring crimp while a third crimp makes sure of the contact between the staple and the wall of the blasting cap. Any static discharge which might pass along the black powder core, will be shunted to ground via the staple and the cap wall. The static discharge cannot come in contact with the sensitive explosive charge in the blasting cap and, therefore, will not be able to initiate the blasting cap. The staple does not interfere with the burning of the black powder core.

Electric blasting caps

Three types of electric blasting caps are used in mining operations:

1. Instantaneous: No. 6 strength;
2. Millisecond delays (short-period caps): No. 8 strength;
3. Long delays (LPV delays): No. 8 strength.

These are all precision-made products with a high degree of timing accuracy. The legwires are plastic coated and closure crimps provide excellent water-resistance. All legwires are factory shunted for protection against static discharge or extraneous current.

Millisecond delays have a nominal delay interval of 25 milliseconds and are available in periods 0-30. They are used in all types of stoping operations and surface bench blasting.

Long delays have a nominal delay interval of 500 milliseconds (1/2 second) and are available in delays 0-15. These are used in development headings, tunnels, raises and shafts.

Non-electric delay detonators

An unusual system of non-electric delay detonators, in millisecond and long delay timing, was developed to provide bottom initiation in holes pneumatically loaded with AN/FO. These ANODET delays are completely immune to the hazards of static electricity or stray currents.

The ANODET delay blasting system consists of a special, high strength delay cap crimped onto a length of 4 grain per foot, low-strength Anoline detonating cord. A special J-shaped connector made of plastic is attached to the ANOLINE to facilitate rapid, tight connexions to a detonating cord trunkline. A further accessory, a plastic cap holder, designed to slip easily over the cap shell, anchors the cap centrally in the borehole so that the AN/FO can be blown past it in one operation.

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The ANODET delay blasting system is widely used in wide cut-and-fill and shrinkage stopes drilled off with either horizontal or up holes, in longhole blasting, sill pillar removal, sublevel drifting and development operations.

TOE-DET delay system

There are few, if any, open pit mines in Canada that use electric blasting caps in primary blasts. Detonating cord systems are used, almost exclusively, to eliminate hazards of lighting and stray currents through conductive ores that might originate from electrically powered shovels and drills.

Detonating cord downlines, however, introduce some undesirable features in bench blasting. Initiation is from the top of the hole down. Analysis of high speed motion pictures shows that top priming increases flyrock, reduces confinement prematurely and contributes to back break. All bubble-sensitized slurries are affected to a degree by the detonation of the downline prior to the initiation of the slurry column. AN/FO too may suffer partial desensitization in the area surrounding the detonating cord.

To minimize these effects the principle of the ANODET delay was applied to develop a more rugged set of components that could be used with cast primers in all open pit mines. This development resulted in the TOE-DET delay system which involves three major components. The TOE-DET delay is a special high strength delay cap crimped onto a length of reinforced ANOLINE detonating cord. This reinforced grade is built up from the original ANOLINE with extra fabric covering for increased abrasion resistance and tensile strength. These delays are available in millisecond timing in a full range of zero to 30 delays. The reinforced ANOLINE is coiled in a figure-8 fold in a variety of lengths as a special "J" connector is used to attach each to the initiating trunkline.

The TOE-DET primer is a specially developed Procore type having TNT cast around a central pentolite core. This primer weighs 1 1/2 lb. A plastic lined tunnel through the centre core permits a TOE-DET delay and reinforced ANOLINE to be threaded through. A cap well on the under side of the primer accommodates the cap within the central core. Two other unlined holes run through the primer to allow an up-line to be threaded through and secured.

Trunkline detonating cord is designed specifically to provide positive initiation of the reinforced ANOLINE. It has a low charge-to-textile ratio for donor strength and excellent knot-tying properties.

The principle of operation is that the TOE-DET delay system picks up its detonation impulse from the trunkline detonating cord. The impulse is transmitted through the reinforced ANOLINE with no effect on the surrounding explosive column. This impulse ignites the delay element in the TOE-DET delay. On detonation this fires the TOE-DET primer which in turn detonates the main column as well as initiating the up-line where used. Several configurations within the borehole as well as in the surface trunkline configuration are possible. When used in conjunction with surface delays, it is possible to have the entire surface trunkline and delay system function before the first hole detonates. This eliminates any damage to surface lines from premature or excessive up-lift of the rock mass.

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Detonating cord

As previously mentioned detonating cord is used exclusively to initiate borehole charges in all major open pit mines. It is also used extensively to initiate secondary blasting charges because of the case of handling and immunity to extraneous currents. In underground operations cords are used for secondary blasting mainly due to electrical safety and excellent impact sensitivity where rock falls may occur in chutes, ore passes or draw points.

As with all Canadian products, cords are designed to function and handle over a broad temperature range particularly in sub-zero weather. At low temperatures they must retain good flexibility and knot-tying properties.

The properties of the six major types are summarized below.

	Nominal weight of explosive (pounds/1,000 ft)	Diameter (inches)	Tensile strength (pounds)
B-line	3.6	0.130	200
E-cord	3.6	0.160	250
Trunkline	7.15	0.176	250
Twin-lead	6.15	0.255	275
Reinforced	6.15	0.200	275
Scuf-flex	7.7	0.237	300

Detonating relays

Millisecond delay intervals can be incorporated into surface trunkline systems through the use of detonating relays. These are two-directional units consisting of a central metal sleeve incorporating the delay elements and booster plus 18-inch long detonating cord pigtails at each end. These are available in standard delay times of 5 ms., 10 ms., 15 ms., 25 ms., 50 ms. Intermediate times and extended delay series are available on special order.

Underground explosives

All explosives used in underground mines must conform to the Fume class 1 rating established by the federal Government. The explosive must produce, on detonation, no more than 0.16 cu ft of total toxic gases when packaged in a standard 1 1/4" x 8" cartridge, or equivalent, as measured in the Bichel gauge.

The principal products and methods of initiation used in the phases of mine development and operation are summarized.

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Shaft sinking

Although water conditions are usually severe, the short exposure time of the loaded explosives permits extensive use of economical semi-gelatin explosives. Ammonia gelatins are used if high water resistance and better tamping properties are required. Higher velocities of detonation and subsequent brisance may be of value in hard ground.

By law, electric blasting caps are required and long delays or selected intervals of millisecond delays are used to control muck throw.

Tunnelling and contract development headings

Generally, high explosives are used in contract operations owing to reliability under difficult conditions of high speed advance. Higher strength semi-gelatin explosives are used but some operators favour ammonia gelatins of up to 80 per cent strength for maximum energy utilization particularly, where small-diameter holes are used. Initiation is by long delay electric blasting caps or these in combination with millisecond delays in larger headings.

In normal mine development headings and raises AN/FO, pneumatically loaded is used wherever water conditions permit. In wet areas or submerged lifter holes, semi-gelatin high explosives are used. Initiation and timing is usually with safety fuse assemblies, Ignitercord and high strength caps which, with pneumatically loaded AN/FO, require no additional boosters. Semi-gelatin explosives are usually used as boosters with EB caps in conjunction with AN/FO.

ANODET delays are used extensively in larger diameter headings particularly where complex Ignitercord circuits may invite trouble. They are also used in horizontal breast stopes and cut-and-fill stopes involving up-holes drilled from mobile rigs.

Longhole stopes and longhole pillar recovery operations which involve bit sizes from 1 5/8" to 2 1/8" and inclined holes up to 100 ft long use pneumatically loaded AN/FO in dry areas with semi-gelatin or ammonia gelatin explosives in wet areas. Initiation is by millisecond EB caps or in some cases ANODET delays.

AN/FO in underground mines

For underground use in oxygen balanced 94 per cent-6 per cent mixture of porous prills and diesel oil is used. The oil is coloured with a distinctive dye so that the mixture can be identified as explosive material. A recent improvement is the addition of a liquid odour suppressant to reduce the unpleasant smell of diesel oil in confined underground workings.

The product is packaged in 50 lb multi-wall paper bags having a special inner liner to minimize oil vapour loss and an outer plastic coating for weather and moisture protection.

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As in open-pit mines the addition of powdered aluminium has proved extremely successful. To date only the mid-range of metal percentage has been used. There are no problems with pneumatic loading and performance in all applications has been equivalent to ammonia gelatin dynamites of 75 per cent cartridge strength. The combination of powdered aluminium with the high densities achievable through pneumatic loading has promising potential.

There are three basic types of pneumatic loaders used underground.

(a) Open cannister, Venturi ejector:

Small diameter holes (1 1/4" to 1 5/8") up to 20 feet long;

Drifts, raises, breasting and overhead stoping.

(b) Pressure vessel, Venturi ejector:

Intermediate diameter holes (1 1/2" to 1 7/8") up to 20 feet long;

Large headings and tunnels, intermediate long hole stopes.

(c) Pressure vessel:

Hole diameters from 1 3/4" to 3" up to 110 feet long;

Longhole stopes, pillars or large tunnel headings.

By law, all pneumatic loading devices must be properly grounded and must use only approved semi-conductive loading hoses. The use of electric blasting caps in conjunction with pneumatically loaded AN/FO is subject to the approval of provincial authorities. Approval is usually given if on-site measurements confirm that equipment groundings, rock conductivity, atmospheric humidity and stray current hazards are within the required limits.

Cap-sensitive, small diameter slurries

It was mentioned earlier, in section A, on classification of explosives, that the two major classes of slurries, self-explosive and non-self-explosive sensitized, were blasting agents. These are not sensitive to initiation by a single No. 8 strength blasting cap. They require high energy boosters and will not propogate in columns less than 3" in diameter.

In October 1971, a third class of slurry explosives was introduced to the Canadian market that is expected to have major impact on the mining industry. These slurries have two drastically different properties to their predecessors:

(a) They are sensitive to initiation by a single No. 6 blasting cap or equivalent. They are thus classed as high explosives;

(b) They will initiate and propogate in boreholes down to 1" diameter.

These radically new formulations have remarkable properties, the major one

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being an outstanding measure of safety with respect to impact, heat and friction even though they are cap sensitive. Their characteristics are as follows:

(a) They meet Fume class 1 requirements for underground use;

(b) They initiate and propogate in holes of 1" diameter with standard detonators at rock temperatures as low as +40°F;

(c) They have a high degree of water resistance;

(d) They are completely free of toxic effects during handling and storage. They will not induce headaches regardless of temperature or ventilation;

(e) They generate an extremely low volume of smoke; thus smoke clearing time, particularly in secondary blasting operations, is drastically reduced;

(f) They are available in a full range of strengths to meet all rock and fragmentation requirements;

(g) They are provided in special cartridges for borehole loading and in packages for secondary blasting;

(h) A specialized grade can be pumped into longholes of any angle for longhole or pillar recovery blasting.

The following tabulation shows the results of Fume tests conducted according to the Federal Government standards previously described.

Product	Total toxic gases (cu ft)
Cap-sensitive slurry	0.03
40% ammonia gelatin	0.06
AN/FO (oxygen balanced)	0.06
70% semi-gelatin	0.09
75% ammonia gelatin	0.11
Maximum permissible limit	0.16

This next table shows the results of impact tests conducted on the standard United States Bureau apparatus that allows a 5-km steel weight to fall on a sample placed on a horizontal steel plate. The impact sensitivity of the explosive is expressed as the height of fall of the weight, in inches required to produce detonation in 50 per cent of the drops made.

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Product	Inches of fall of 5-km weight for 50% detonation frequency	Foot-pounds of impact energy
50% straight dynamite	6	5.5
75% ammonia gelatin	20	18
75% "low sensitivity" dynamite	48	44
Cap-sensitive slurry	No detonations to limit of apparatus.	

Cap-sensitive slurries cannot be detonated by high energy rifle bullets. Thermal tests which involve the heat stability and detonation susceptibility as well as ignition from direct flame show that these compositions will decompose at 220°C without detonation. They burn as long as flame is directly applied but cease burning when the flame is removed.

For borehole use, in packages, a unique cartridge has been developed. This is a plastic tube formed by a longitudinal heat seal on layflat film and closed on the ends by metal clips. Newly developed equipment gives a completely filled cartridge with lightly tapered ends that can be loaded intact or slit to improve loaded density.

A high-energy product, sensitive to initiation by a single detonator or low core-weight detonating cord, plastic in nature to provide good contact with irregular rock surfaces and completely non-toxic with respect to headache-producing components, was quickly recognized as "a natural" for secondary blasting. To utilize fully these properties and to provide the maximum area of contact - a long established criterion of secondary blasting - the Pillow-Pak charge was developed. This is a special package available in 1, 2 1/2, 5 and 10 pound sizes. With the exception of the ten-pound size it is available in "strip" packaging composed of four separate units joined by a perforated web. Individual units can be torn off and two or more can be used to increase charge weight. They can be folded one on top of the other or laid side by side provided good contact is maintained. This is a convenient package for pale charges in chutes or ore passes.

The advantages of bulk loading slurries and AN/FO mixtures in large-diameter down holes in open-pit mines are well recognized and established. The same exists with pneumatic loading of AN/FO into small-diameter holes in underground mines.

These advantages are: speed of loading; high load densities; perfect borehole coupling; and reduced manpower.

AN/FO mixtures have serious limitations, however, in wet areas particularly where long exposure times are required such as in pillar blasts. A slurry product having exceptional water resistance, good borehole coupling features, able to initiate and propogate in small diameters and able to be bulk pumped and retained in vertical up-holes was an ambitious objective. This objective was reached. A unique slurry product and bulk pumping system for small diameter holes was introduced to the Canadian market in 1972.

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A specialized pump has been developed that will pump this bulk slurry at rates up to 120 lb/min through hoses up to 400 ft long. This system is in use in underground longhole stoping and pillar blasts and shows great potential in surface quarry operations, pipeline trenching and construction projects. The pump is a double acting, piston type powered by a 60 to 100 psi air supply. The pump itself is 12" long, has a cylinder diameter of 4 3/4" and a total displacement of 0.02 cu ft per stroke. This pump has been designed for safe operation. Balls and ball seats are all nylon, sleeves and sleeve packings are nylon and leather and there are no metal-to-metal parts.

Auxiliary equipment involves a hopper mounted above the pump intake. This hopper is of sectional construction that allows capacities of 60, 180, 240, and 480 pounds. For underground use the entire unit is skid-mounted with dimensions, exclusive of the detachable hoppers, of 72" long x 20" wide x 23" high. Total weight is 225 pounds. It can be handled easily by two men and can be transported through a mine system with no dismantling required. Smaller units can be made for special applications and the design is easily adapted to wheels or vehicles of any type.

A new, hydraulically powered design is nearing completion for special application to mobile equipment, particularly where compressed air may not be available.

The safety features of this system are, to a large extent, due to the properties of the slurry itself. It is sensitized by micro-bubbles in conjunction with fine aluminium. It was stated earlier that this "sensitizing mechanism" is pressure sensitive as the energy of adiabatic compression is less if the bubble volume has been reduced by pressure prior to initiation. This slurry is "dead pressed" at 30 psi and at this pressure cannot be initiated by even 100 grams of pentolite. Pumping pressures may run as high as 200 psi depending on the length and diameter of hose, thus the product is totally "dead pressed" during the pumping operation. The gel consistency has elastic properties that allow it to recover its normal cap sensitive density within two to three minutes after the pressure is released. The system is soon to be introduced into surface quarry operations using holes of 2 1/2-4" diameter. Successful trials have been conducted on a large pipeline project using 2 1/2" diameter holes. In these two applications the high load densities, excellent coupling have allowed substantial increases in pattern plus additional savings in reduced manpower.

An example of the underground application was the use of this system to solve a major problem in a pillar blast involving 182,000 tons of ore. This pillar was drilled off well in advance of loading as is the usual practice. Ground movement during the pre-blast period resulted in the partial blockage of 200 of the total of 600 holes which were 2 1/8" diameter drilled to any average depth of 50 ft from horizontal to vertical in standard rings.

As these holes could not be loaded with cartridged explosives and water conditions prevented the use of pneumatically loaded AN/FO, the only choices were to leave the holes empty and risk disaster or to re-drill approximately 6,000 ft of holes. By pumping this new bulk slurry through combinations of 1" and 3/4" delivery hoses all of these holes were loaded successfully and the pillar was

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blasted on schedule with excellent results. In most cases, it was necessary to use 3/4" diameter hose to get past obstructions in the holes. Twenty-one thousand pounds of slurry were loaded - just over 33 per cent of the total load.

In addition to the fact that these holes could not have been loaded by any other means, at least one week of loading time was saved with much reduced labour force. The bulk slurry was loaded at a rate of 2,540 lb per man shift compared to 250 lb per man shift loading cartridged high explosives.

A brief prediction of the future is appropriate. Mechanical loading of explosives into even the smallest diameter hole will be perfected. Products of the required consistency and safety to apply to mechanical loading will be developed with the greatest emphasis on bulk loading systems for all applications.

Basic ammonium nitrate mixtures will see more extensive applications. The full potential of this cheap and readily available chemical has not yet been reached. Increased densification through chemical and mechanical means and more sophisticated liquid sensitizers will produce borehole densities and energy values equivalent to most slurry explosives in use today. Borehole dewatering and lining techniques will be further improved to extend the use of these blasting agents. Metal additives will see greater use in all dry blasting agents. The past decade of explosives technology has seen truly remarkable changes. The future decade will be a continuing and probably accelerated environment of change and progress.

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III. MINE ENVIRONMENTAL ENGINEERING

M. J. McPherson*

In recent years there has been growing public awareness of the effects of industrial growth on the general environment. Attention has been focused upon pollution of the atmosphere, oceans and rivers and despoilation of the land. Forecasts have been made which indicate that unless ecological consequences are taken into account in the planning and operation of present and future technological enterprises then the quality of human life will be most adversely affected.

The necessity for control is no less acute within the industrial environment itself. In the mining industries of the world, minimum standards of acceptability are laid down by national mining law. Such legislation is tending to become more stringent. This is due to the public in general and the mine workers in particular becoming increasingly conscious of the physiological and psychological dangers associated with employment in the extractive industries - in both the short and long terms.

A situation exists in which the production from many mines will depend to a large extent on the ability of mining engineers to maintain environmental control in and around the workings. It follows that those concerned with the mine environment must be closely involved in planning procedures. This paper outlines a number of environmental hazards which may arise in mine workings, together with some of the factors which should be taken into account in planning means to combat them.

A. The mine environment

The specialist engineers who are responsible for providing acceptable working conditions in mines must necessarily have an understanding of those parameters which affect the health and efficiency of mine workers. These include dust, gases, heat and humidity, noise, lighting and radiation. Furthermore, the mine environmental engineer must have a knowledge of the techniques available for control of the climate in mines.

The over-all requirement is that the air should be kept within such limits of purity, dust contamination, temperature and humidity as to enable men to complete their tasks in reasonable comfort and with minimal effect on their health.

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Dust

The effect on health. The several lung diseases caused by over-exposure to dusts are known by the generic name "pneumoconiosis" (alternatively pneumonokoniosis). This has been defined formally as "the accumulation of dust in the lungs and the tissue reactions to its presence". All too frequently associated with tuberculosis, these diseases have been known since the earliest days of mining. Unfortunately, mechanization and modern extraction methods have exacerbated the problem.

The precise effect on the lung depends upon the type of dust. In general, the development of fibrous and scar tissue reduce the elasticity and effective surface area, resulting in reduced gas exchange across the internal lung surfaces. The important parameters are the chemical nature of the dust, particle size, concentration and exposure time.

Limestones do not give any specific form of pneumoconiosis. However, siliceous impurities can make such dust dangerous. Silica dusts are the most hazardous - the hard angular particles cause severe scarring of the lung tissue resulting in silicosis. Asbestos is a fibrous silicate which causes abestosis and is also associated with forms of lung cancer. Coal dust not only causes scarring but tends to be retained by the lung giving a progressive accumulation of carbonaceous material within the tissue.

It has been stated that particles of five microns and below are those which give rise to pneumoconiosis. The larger particles are filtered out in the upper respiratory system. However, these larger particles lead to other problems associated with vision and the subjective feeling of comfort. The mining engineer has a responsibility to ensure that dust concentrations are maintained at an acceptably low level and that workers are not exposed to high concentrations for long periods.

The onset of pneumoconiosis can be recognized at an early stage. The disease cannot be cured but it can be controlled to a certain degree. It is, therefore, in the interests of mining industries to ensure that their permanent labour forces are examined for pneumoconiosis at regular intervals of time.

Production of dust. Whenever mineral is broken by impact, abrasion, crushing or by the use of explosives, then dust is produced. As a generalization, a high dust level indicates an inefficient use of energy. Many of the measures taken to reduce dust levels are no more than good mining practice.

In a short time immediately following rock breakage, some 95 per cent of the respirable dust (below five microns) aggregates together. This process is assisted by the humidity of the air, the application of water at the point of breakage and by a low air velocity.

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The effect of air velocity on airborne dust concentrations depends to some extent on the source of the dust, that is, whether it is on a stationary surface, falling freely or moving due to a mining process. Figure 8 shows that respirable dust can be diluted by increasing the relative air velocity to approximately 1.5 m/s (300 ft/min). At higher air velocities, dispersion of the dust occurs, commencing with the larger particles. At air velocities above 2 m/s (400 ft/min) the respirable dust level may begin to increase significantly. As a general recommendation, the relative air velocity over mineral winning or transport operations should be maintained within the range 1.5 m/s to 4 m/s (300 to 800 ft/min). In some situations, however, this upper limit may have to be exceeded for other reasons such as heavy gas emissions.

Mining machinery and dust suppression. The most prolific sources of dust in mining are usually at the points where the mineral is being broken from the solid. Where rock breaking machines are employed (drills, power loaders, tunnelling machines), much can be done to minimize dust emission. The proportion of dust produced may be reduced by increasing the top size of the broken rock. The major causes of dust from rock breaking machinery are blunt picks, shallow cuts and abrasion due to inadequate clearance. It follows that well maintained, slow-moving cutting heads will make most efficient use of input power and hence reduce dust (figure 9).

The use of water is commonplace on drills. On larger rock breaking equipment, the design and position of the sprays are most important. The correct application of water at the point of impact (pick face flushing) can be most effective in reducing airborne dust. After respirable dust has become airborne it is difficult to remove by conventional water sprays. The air velocity over moving picks can be reduced by the use of cowls.

Where explosives are used to break the mineral, it is important that the most effective drill hole pattern and charge per hole is determined - and then adhered to. The use of water/gel ampoules for stemming also plays a part in reducing dust and shotfiring fumes. Fog sprays switched on before firing and left to run for some period afterwards may be effective, particularly in hard-rock mining and in tunnelling.

In some mining situations water infusion of the mineral prior to working has proved effective. Mainly employed in coal mines, the technique involves drilling holes into the area to be worked and injecting water under pressure. It has been shown that within a given coalfield, the higher the moisture content of the coal, the lower is the incidence of pneumoconiosis. The length of infusion holes and the water pressure to be used vary with local conditions. The success of this technique depends upon the permeability of the mineral in situ. Permeability tests within the rock body should precede the full-scale installation of water infusion equipment.

A source of dust which is increasing on mechanized longwall faces is the

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Figure 8. Effect of air velocity on pick-up of dust

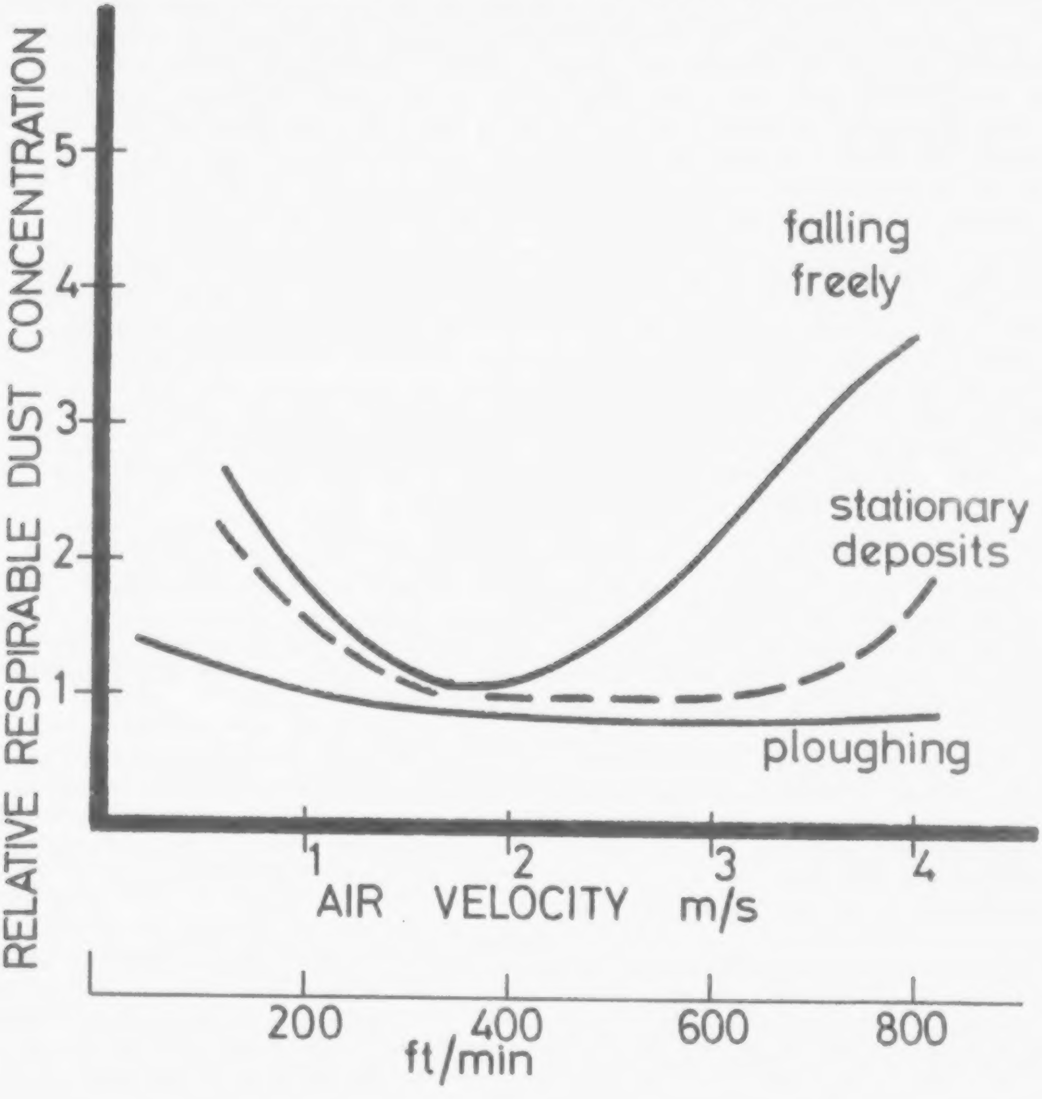
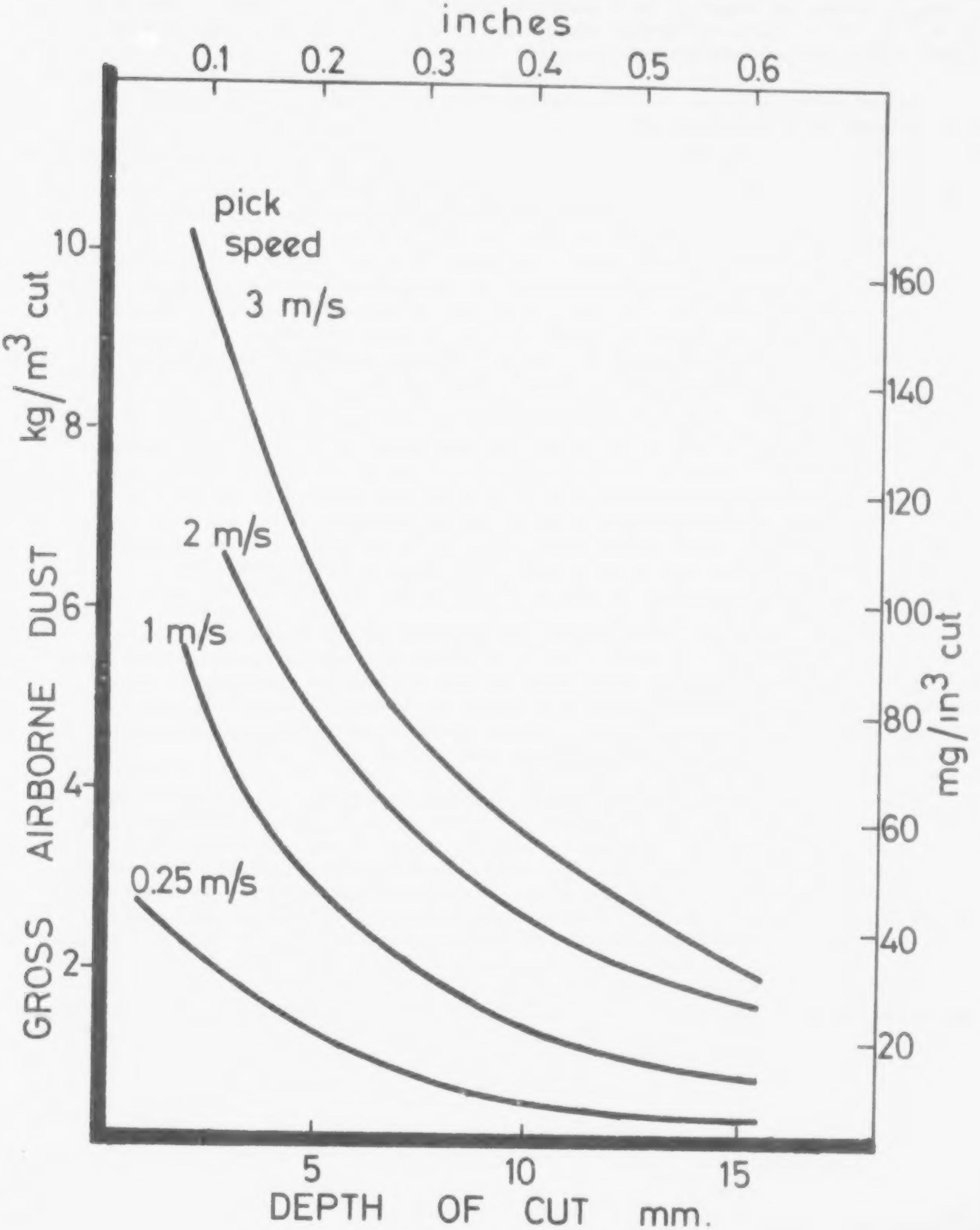


Figure 9. Influence of pick speed and depth of cut to airborne dust



crushing of roof and floor strata by powered supports. As the setting and yield loads of hydraulic supports increase, so does the amount of powdered material produced. This may be minimized by the use of cushioning; paper has proved quite effective in British coal mines.

Moving away from the mineral winning areas, dust is produced at loaders, crushers and at numerous points along the mineral transport system. Dust suppression at loaders and crushers can be facilitated by strategically placed sprays and, where practicable, by the use of shrouds. It should be emphasized that the most important parameter when using sprays is their position. The effectiveness of a badly placed spray is not significantly increased by increasing the water flow.

During transport of the mineral, the aim should be to prevent drying of previously wetted material. In some cases, market restrictions will determine the amount of water which can be added to the mineral en route. The sites of dust production during transport are at loading and unloading operations, conveyor transfer points, bunkers (particularly at discharge), skips, airlocks and vehicle arresters.

During transfer of the mineral by gravity, free fall should be avoided wherever possible. Chutes should be designed at an angle slightly less than the repose angle of the broken mineral. Shrouds should be used at gravity feeds. Where the mineral transport is in intake airways, it is sometimes possible to run an air duct from shrouds to a return airway. This utilizes the mine ventilating pressure to give a cheap dust extraction system. In other cases mechanized dust extraction equipment may prove to be necessary.

On conveyor systems, rewetting of the mineral should take place by a low-pressure "dribbler" bar spray some 10 m before a transfer point. Joints on conveyor belts should be made dust proof by vulcanizing, insert strips or by sealing compounds. Particular care should be taken when conveyors are required to pass through airlocks. In such situations, consideration should be given to shrouding the relevant length of conveyor.

Vehicle arresters are often unnecessarily severe in giving shock loads. This is not only bad for the equipment but also produces dust. Deceleration devices can alleviate such situations. Maintenance of transport equipment can do much to prevent unnecessary dust. Tracks should not be allowed to develop sudden changes of direction or gradient. Conveyor structure must be cared for by the replacement of damaged rollers and attention paid to belt alignment, centring devices and bottom belt scrapers or brushes.

At major mineral transfer points such as at shaft bottoms or where shrouding is difficult, thought may be given to using an additional ventilation

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split to bypass the majority of the airflow. Another method of reducing the relative velocity between airflow and mineral transport is to employ homotropal ventilation, that is, mineral transport in the return airways. This not only reduces dust pick up during transport but also ensures that the intake air reaching the working areas is relatively dust free. However, in some countries, mining law effectively prohibits this practice.

Other techniques of roadway dust suppression include moist impingement curtains. These are effective for the non-respirable dust and restrict the airflow. However, they can prove useful when erected temporarily for shotfiring. Hygroscopic wetting agents may be employed to prevent settled dust from again becoming airborne. Calcium chloride crystals remain effective for four to six months. Sodium chloride can also be used but only when the air relative humidity remains above 75 per cent. When using chemical compounds for roadway dust consolidation, care should be taken to prevent corrosion of tracks or other floor-mounted equipment.

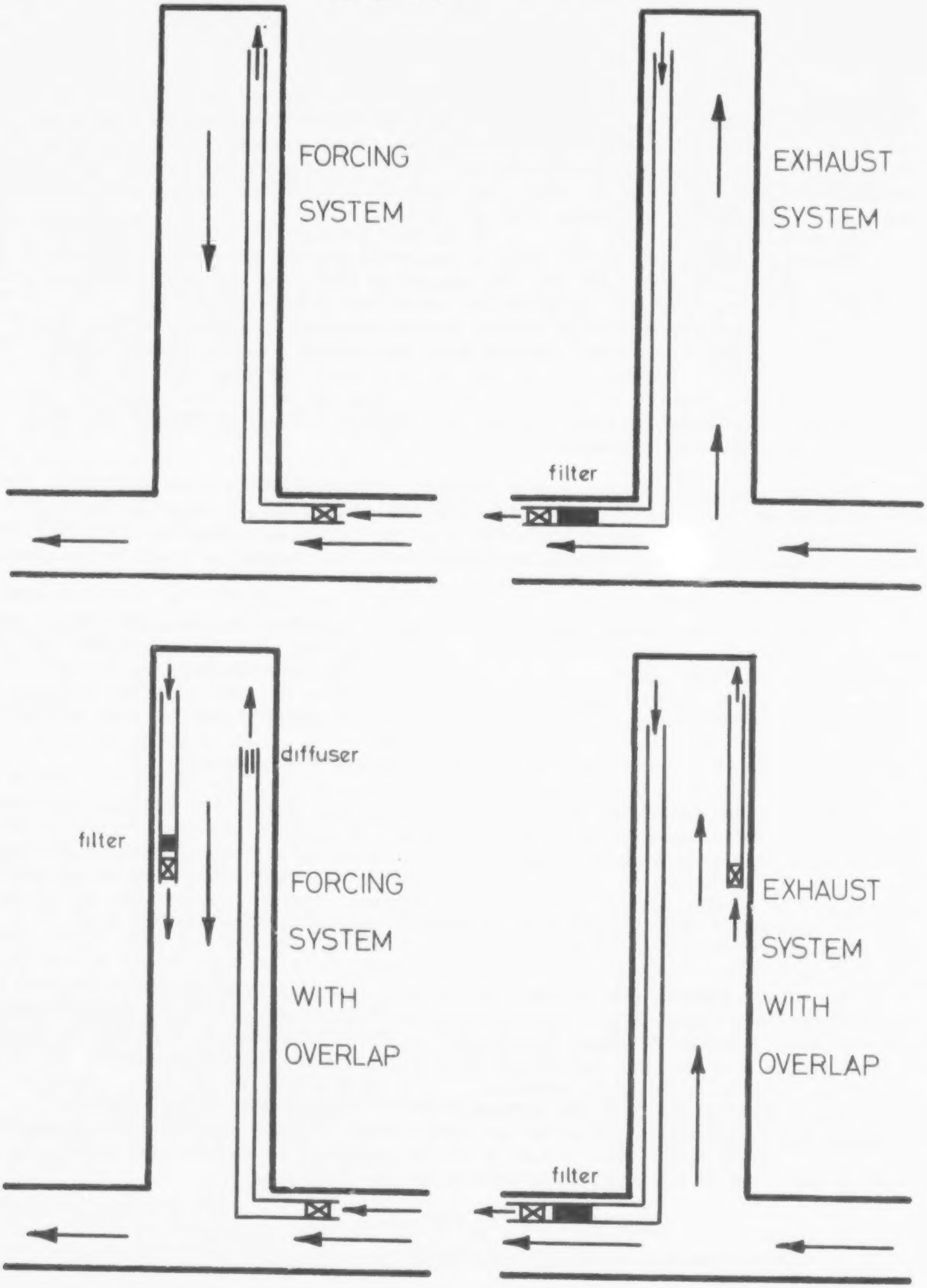
Underground tunnelling and dust filters. Dust-suppression techniques in the driving of tunnels merit special attention. The dust-emission pattern in such cases is generally heavy concentrations for relatively short periods of time. In addition to the methods outlined above, the method of ventilation should be considered from the dust viewpoint.

Figure 10 shows typical air-duct layouts. A conventional forcing system, on the one hand, ensures that the high dust concentrations will be maintained for the maximum time and that all the men in the tunnel will be affected. On the other hand, a completely exhausting system may give problems of poor scouring of the face in gassy conditions. An overlap system of air ducts gives the best of both worlds. The open end of the exhaust duct should be as close as practicable to the face of the heading in order to achieve maximum dust capture.

Dust filters should be incorporated into exhaust ducts. It is important that the correct combination of fans, duct sizes and filters be chosen for any given duty. In some cases, it may be necessary to employ an additional fan, connected in series, to overcome the pressure drop across the filter. Three types of dust filter are in common use in the mining industries: fabric filters, wet scrubbers and wet spinners. Electrostatic precipitators may be considered also for non-gassy atmospheres.

The fabric filters are the most efficient, removing some 90 per cent of respirable dust. They have, however, a short life and offer a varying resistance to airflow. The wet scrubbers (Microdyne Type) introduce water into the air/dust mixture via a wire screen, the wetted dust being removed by impingement on louvres. Their efficiency is normally in the range 60 to 80 per cent. Little maintenance is required and the equipment has a long working life. The wet spinner filter introduces water to the centre of a spinner cap which forms part of the boss on an axial fan. Water is thrown outwards by centrifugal action and some 50 per cent of the dust is entrained in the droplets.

Figure 10. Duct systems for auxiliary ventilation of tunnels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In the interests of economy, the filter chosen for any given installation should depend upon the subsequent path of the filtered air. If the auxiliary ventilation is handling intake air which will be used again in other working areas, then a good dust filter should be chosen. If, however, the air duct is exhausting directly into a return roadway, then a less efficient filter will suffice.

Little can be said here on the important topic of airborne dust sampling. Many instruments have been designed and are in use for this purpose. As a measure of liability to pneumoconiosis, the sampling device should be selective in the particle size and should be of a long-running type. As exposure time is so important (in addition to dust concentration and particle size) it follows that spot samplers give poor correlation with the health hazard. Selective gravimetric samplers running for a full shift are now regarded as a more accurate indication of the dust environment.

Mine gases

Occurrence and associated dangers. In addition to the normal constituents of air, there are a number of gases which may be found in underground workings. The more common of these include methane, carbon dioxide, carbon monoxide, nitrogen and sulphur oxides, hydrogen, hydrogen sulphide and radon. Each of these gases has its own associated hazards. However, even in the absence of emitted gases, stagnant air underground is liable to become incapable of sustaining life owing to depletion of the oxygen content.

The most common mine gas is methane. This occurs to some degree in all coal mines although coalfields vary widely in their "gassiness". It can also be emitted in non-coal workings which underlie or overlie carboniferous strata. This occurs, for example, in the Orange Free State goldfields of South Africa - at Virginia Gold Mine drained methane is utilized to supply a surface steam-raising plant.

The major danger associated with methane is its flammability in the range 5 to 15 per cent when mixed with air, being most explosive at approximately 9.5 per cent. As it is considerably less dense than air, methane tends to accumulate in the roof of workings and form long streamers or layers. In the event of an ignition such layers behave as a fuse leading to larger accumulations. Ignitions of methane are a danger to life by burning, blast, asphyxiation and poisoning from the production of the highly toxic carbon monoxide. Even more serious, however, is the possibility that the shock wave may raise sufficient concentrations of dust to propagate a coal-dust explosion with disastrous consequences.

Carbon dioxide is produced by the oxidation of organic material. It can also occur as a strata gas. Although it has a relatively low toxicity it can become asphyxiating owing to the displacement of oxygen. Carbon monoxide occurs due to the incomplete combustion of methane, coal or other organic compounds such as petroleum fuels. It is a highly toxic gas and can lead to severe loss of life following explosions or underground fires.

The oxides of nitrogen and sulphur are also toxic and may be produced by blasting, internal combustion engines or the heating of certain ores. Hydrogen and hydrogen sulphide may occasionally appear as strata gas. More usually they are associated with acidic mine water. Hydrogen is also a hazard at battery charging stations. Radon is a particular hazard in uranium mines and is a source of radio-active emission.

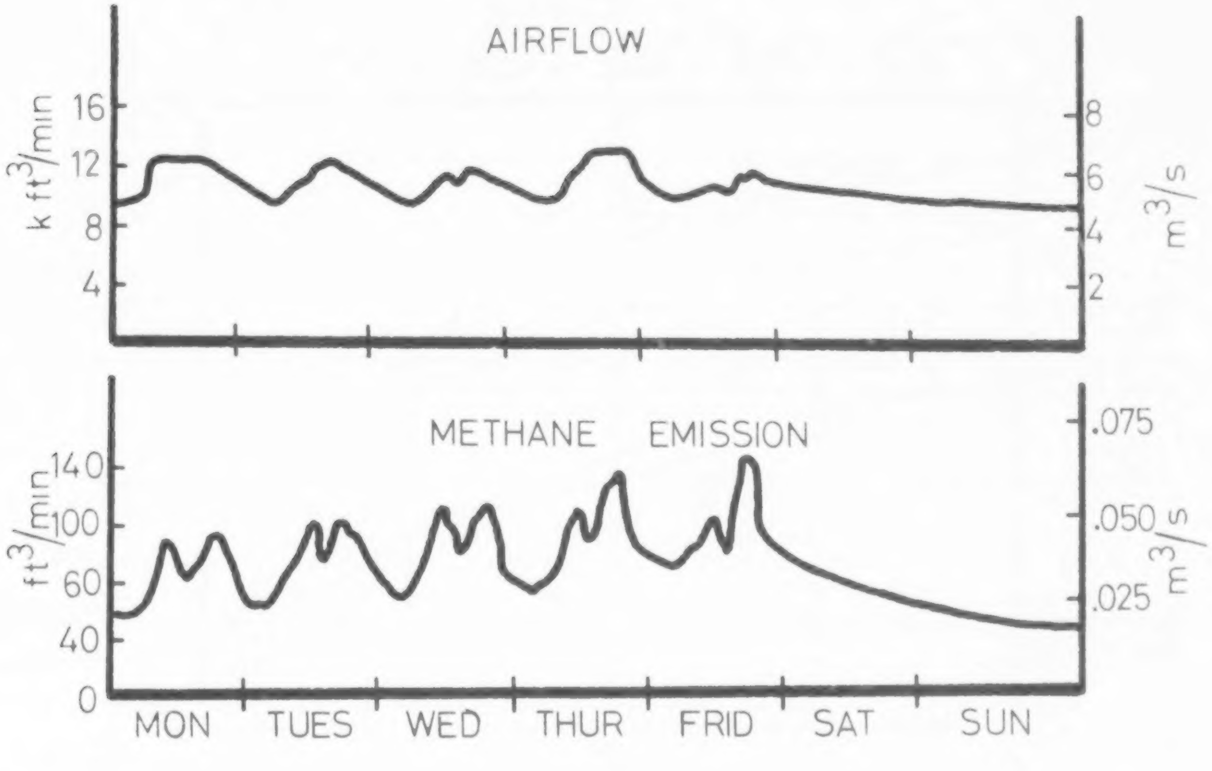
The release mechanism of strata gas. The most prolific sources of mine gas are the strata in which mining excavations are made. A great deal of research has been carried out over the years to gain an understanding of how gas is entrained within the strata and the factors which govern its migration towards mine workings. In general, the gas is held within the rock matrix at a pressure higher than that of the atmosphere. The introduction of mine openings sets up a gas pressure gradient within the strata. Gas will therefore migrate towards the workings at a rate determined by the pressure gradient and also by the permeability of the rock.

In practice, the situation is often complex. Most of the gas may not be in the gaseous state at all. This is the case in coal seams where the greater proportion of the methane is adsorbed; that is, it is deposited as molecular layers on the surfaces of the internal pore structure of the coal. Thus, far greater volumes of methane will be entrained than simple gas-pressure measurements in boreholes would suggest. Furthermore, the rock permeability varies not only with direction - it also changes with time owing to the mining process causing stressing and subsequent relaxation of the strata.

The source of the strata gas may be the mined mineral or the native rock (or both). To establish safe monitoring schemes, it is important to appreciate the variations in gas emission and how these are related to mine production cycles. Where the major gas source is the mineral itself then there will be a direct relationship between mine production and gas emission. Figure 11 shows a smoothed methane-emission pattern from a colliery longwall district employing a power loader on two out of three shifts per day. In such cases, the breakage of the mineral gives high concentrations of gas at the points of comminution. Most methane ignitions are initiated at the machine pick points. Where the larger proportion of gas is emitted from super- or sub-adjacent strata, the relationship between rate of mineral production and gas emission is much less direct. This is due to the time dependence of the redistribution of strata stresses. The layout of workings, methods of strata control and the geology of the area then take on far greater significance.

A measure of gas emission in common use is the volume of gas per ton of mineral produced (m^3/ton or ft^3/ton). Whilst this is a useful and normalized concept for comparative purposes it can be misleading where strata are of high

Figure 11. Cyclic pattern of airflow and gas emission



permeability or the majority of gas comes from sources other than the worked mineral. An empirical relationship of the form

$$E \text{ proportional to } P^n$$

is sometimes used, where

E = gas emission in volume/unit time

P = rate of mineral production (tons/day)

and n = has a value determined by the area and the mining conditions.

The value of n may vary from 0.2 to over 1 in exceptional cases. However, for planning purposes, it is preferable to employ actual gas emission values which have been measured in the same or a nearby mine.

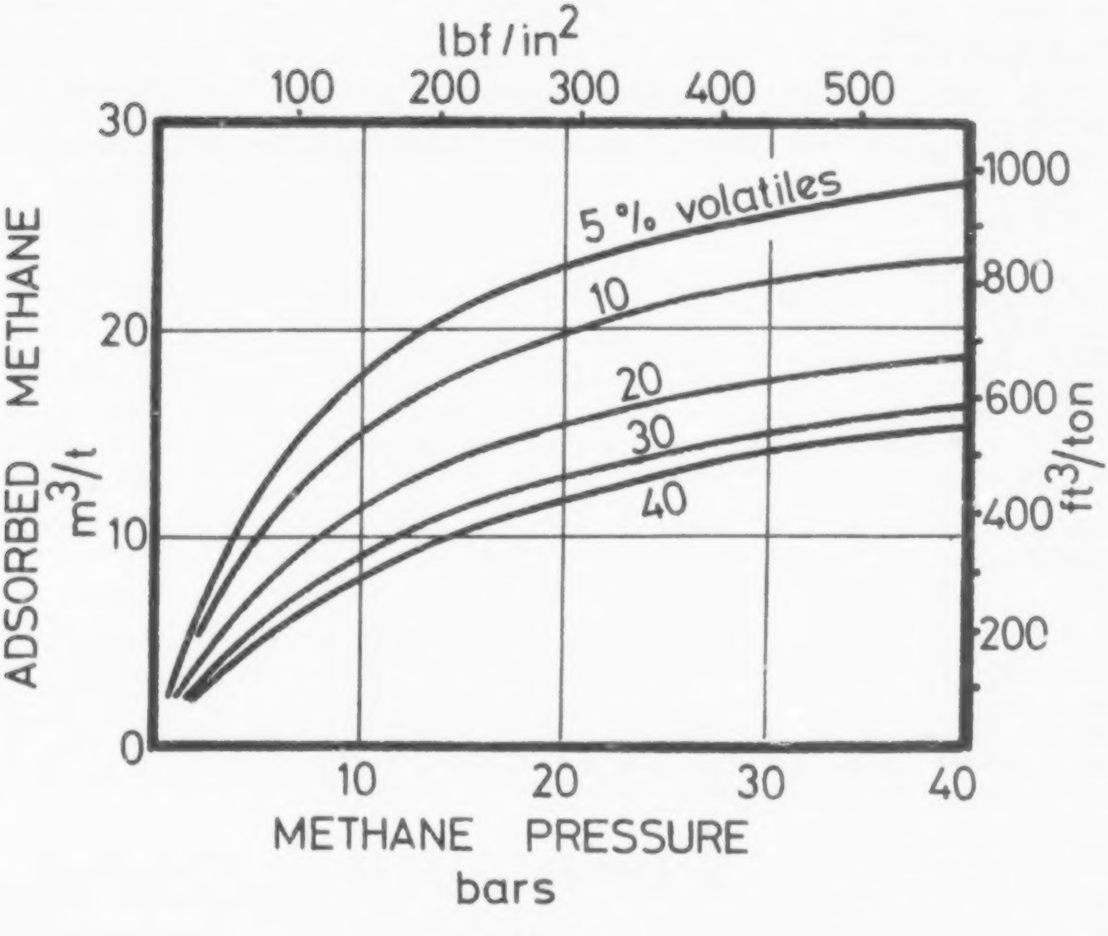
Where it is required to determine the gas content of a stratum, two methods are available. Boreholes are drilled into the rock to a distance beyond the effect of mining stresses. The first method involves taking a core sample or drill chippings and placing them in a hermetically sealed container. This is transported to a laboratory where the mineral is broken down to a specified particle size. The volume of gas emitted during comminution is measured and corrected for loss during drilling. The second method involves sealing the borehole and measuring the gas pressure which is built up. Laboratory pre-determined adsorption isotherms for the mineral at the appropriate temperature and moisture content can then be used to read off the maximum amount of gas that the strata can hold. Typical adsorption isotherms for methane in coal are shown in figure 12. From such information, together with the natural or induced permeabilities the probable gas emission into the mine workings can be determined.

A particular gas hazard which exists in some mines is the sudden emission of large volumes of gas. The expansion of gas may be violent and sometimes associated with the expulsion of solid material. The study of such phenomena is highly specialized, as many interacting variables are involved. Tests have been devised which utilize both steady state and transient measurements of gas pressure and flow in exploratory boreholes.

The major factors which govern the amount of strata gas removed from mine workings are type of strata, depth, current and past geology including tectonic stress history, method of mining, sequence of extraction and rate of production.

Monitoring and control of mine gas. There are a large variety of devices and instruments that have been devised for measuring gas concentrations in mines, ranging from chemical detector tubes to integrated electronic monitoring systems. The minimum amount of routine testing for particular gases may be specified by mining legislation. From the technical standpoint the cost and sophistication of a gas monitoring system should be related to the hazard. In the simplest case,

Figure 12. Sorption isotherms for dry coals (Mücke)



measurements are made at set intervals of time and at specified places by hand-held instruments.

Continuous recording instruments have had a great impact on the understanding of gas emission patterns and have prompted further research into the problems of dangerous peak concentrations which occur during mineral winning. Such instruments are normally placed in return airways and may record gas concentrations locally or at some remote central station. Visual and/or audio alarm facilities are provided should the gas percentage exceed a pre-set level. Alarm indicators may also be mounted on individual machines.

In addition to giving immediate and permanent records of gas concentrations, continuous recording is also most valuable for the detection of a slowly developing hazard. For example, computerized trend-analysis techniques applied to recordings of carbon monoxide can highlight the onset of spontaneous combustion long before any other indication.

The control of gases starts at the mine-planning stage. There is a close link between strata control and gas emission. The mining methods used and the layout of workings have a great influence on the pattern and level of gas emission. For a given layout, the first line of defence against gas is dilution by the airflow. The fans chosen should be capable of producing enough airflow to reduce gas concentrations to harmless levels in all working parts of the mine. It is, therefore, important to be able to predict gas emission rates and hence determine the fan requirements. The cross sectional areas of underground airways are chosen on the basis of a number of factors including cost of excavation, installed equipment, volume of airflow and air velocity. The latter parameter is of importance not only for dust but also in reducing the tendency for gases to stratify, particularly in inclined airways.

Much can be done to improve gas dilution by good routine ventilation control underground. Excessive air leakage between intake and return airways should not be allowed to develop. Measurements of airflow on reliable instruments should be made at regular intervals and undesirable trends rectified before they lead to trouble. Abandoned areas of the mine should be sealed effectively wherever practicable. Where spontaneous combustion is a problem then sealing off is essential.

Drainage of gas from the strata may be practised where the natural or induced rock permeability is large enough. This has been particularly successful in a large number of coal mines where boreholes under suction capture methane from gas sources in the strata above and below the seam being mined. Many areas of coal would be unworkable at modern rates of extraction were it not for methane drainage techniques. If the drained methane can be maintained within given limits of quantity and purity then it may be utilized in surface power generation plant.

The design of mining machinery should ensure that any exhaust gases produced should be minimal. In particular carbon monoxide and nitrous fumes from internal combustion engines should be held at a low level by using the correct fuel, exhaust scrubbers and by stringent maintenance of the machines. Regular analyses should be carried out on the exhaust gases from each engine. Electrical equipment for use in gassy mines must, of course, be either flameproof or intrinsically safe. In addition equipment such as compressed air devices must be earthed against electrostatic sparking. Machinery developed for mining use must be especially safeguarded against frictional sparking or overheating.

In mines where there is any possibility of a gas or dust explosion, stonedust or water spray barriers should be installed. These are triggered by the initial shock wave of an explosion and result in an oxygen deficient length of airway through which the flame cannot propagate. Rapidly acting water sprays powered by compressed gas are being developed.

Heat and humidity

The cooling power of an environment. A number of factors determine the sensation of comfort experienced by a man in his working environment. Two of the most important in mining are wet-bulb temperature and air velocity. The first of these is governed by the air temperature and moisture content.

The human body is an inefficient heat engine. Energy transfer within the body results in metabolic heat generation. In order to prevent heat stress, metabolic heat must be transferred to the surroundings at least as rapidly as it is produced. Thermal equilibrium is established when the cooling power of the environment equals metabolic heat generation.

Conductive heat transfer from the human body in air is negligible. Convective cooling will be constant for a given air velocity, skin temperature and body posture. This is also small compared with the major effect of evaporative body cooling in hot climatic conditions. The nervous system responds to heat stressing by producing perspiration. This may be continued until the body surface is completely wet when evaporative cooling reaches a maximum. Variations in heat loss may therefore occur by regulation of the sweat rate.

The two methods of measuring the cooling power of an environment which have found most favour over the years are the empirical nomograms of Effective Temperature and the Wet Kata Thermometer. Neither of these indices is completely satisfactory. Extensive experimental and theoretical research by the Chamber of Mines of South Africa has led to the development of Specific Cooling Power, defined as the maximum cooling power of an environment and measured in W/m^2. Although this takes into account all of the relevant variables, realistic values for the less sensitive of these have been chosen for deep mining conditions leaving Specific Cooling Power to be computed or read from charts as a function of wet bulb temperature and air velocity only. Although no general claims are made, Specific Cooling Power promises to be an effective and realistic indication of the acceptability of an environment for working men in relation

Similar work has led to the use of controlled acclimatization procedures. At present there are 25 surface and 7 underground acclimatization chambers in South Africa, holding from 60 to 300 men. The period of acclimatization is three five or eight days, preparing men to work in wet bulb temperatures not exceeding 29°C, 31°C and 33°C respectively. The air velocity in the chambers is maintained at 0.4 m/s (80 ft/min) with the wet bulb temperature between 31°C and 31.7°C and the dry bulb temperature between 33°C and 33.3°C. Initial acclimatization has led to a significant reduction in the incidence of heat stroke in South African mines.

Sources of heat. Detailed lists of the sources of heat and airborne moisture in mine workings are available in mine ventilation literature. Only the most relevant are discussed here with particular reference to modern mining methods.

Although not a source of heat, the effect of air compression is important in downcast shafts or dip workings. The precise rate of temperature rise depends upon the inlet conditions but will be of the order of 0.4°C wet bulb per 100 m depth. The two major sources of heat are the strata and machinery. Both are increasingly troublesome as mine workings become deeper and more intensively mechanized. The deeper mines of the world have had to employ air conditioning for many years. However, cooling plant is now having to be considered for relatively shallow mines which use high-powered machinery.

Heat energy is transmitted through solid strata by conduction. The rate of heat transfer towards the workings depends upon the difference between the virgin rock temperature (VRT) and the temperature of the ventilating airstream - and also upon the thermal conductivity of the rock. The VRT can be assessed from a knowledge of the local geothermic gradient. Care should be taken, however, in the interpretation of average geothermic gradients given for wide geographic areas. There may be significant variations due to localized geology. Indeed, even for a single borehole, geothermic gradients are not always continuous straight lines. Local data should therefore be used to determine VRT for a given mine take. Similarly, thermal conductivity should be determined in situ. Laboratory measurements of small solid samples may give misleading results.

To illustrate the importance of machine heat, a 138 kW diesel loader (185 hp) dissipates virtually all of its power in the form of heat to the surrounding environment. Allowing for machine inefficiency some 60 refrigeration tons must be applied to counteract the effect. For practical calculations, all power which is supplied to machines is utilized either against gravity or appears as heat in the ventilating airstream.

Control of heat and humidity. As with so many mine environmental problems, difficulties arising from high temperatures should be considered in the early stages of planning the mine. The heat and moisture produced by machines are introduced artificially into the mine and are therefore intrinsically more amenable to control than natural sources of heat and humidity. When mechanization is to be installed it is important that motors are not over-rated. The equipment should be powered to no more than the required level. Furthermore, it should be maintained to work efficiently.

The work of machines should be spread over a maximum time period and peak loadings avoided wherever possible. Motors should be run only when necessary. In particular, conveyors should not be allowed to run empty for long periods of time. The use of compressed air equipment transfers the machine heat away from the working area.

Machine cooling is not so straightforward underground as on surface locations. The machine emits most of its heat by convection and radiation. Water cooling necessitates both a water supply and a return circuit in order that the heat may be rejected into return air at some convenient point. Machine cooling water should not be utilized for any other purpose in the working area.

Water sprays used for dust suppression should be carefully maintained and controlled. Machine pick face flushing is much more effective than finely divided sprays. Nevertheless, water used near or at the point of mineral breakage should be kept to the minimum consistent with acceptable dust levels. The temperature of the rock surface is at a maximum here and water evaporation rates will be high.

Fixed motors sited in intake airways can sometimes be shrouded and an air duct connected to a return airway. Provided that national mining law permits, consideration should be given to the implementation of homotropal ventilation. This implies that mineral flow and airflow are in the same direction and that mineral transport is in return airways. Homotropal ventilation may bring other problems associated with drainage, gas control, natural ventilating effects in open stopes and fire fighting. On the other hand homotropal ventilation ensures minimum contamination of intake air by dust, gas, heat and moisture.

The high costs of shaft sinking and subaqua workings have encouraged mining engineers to utilize shafts for increasingly large extraction areas. This requires higher duties of fans and air conditioning plants. Before installing refrigeration equipment, the mine environmental engineer should investigate thoroughly all possibilities of alleviating the situation by modifications to the airflow system. If the problem of high temperatures exists only in positions difficult to ventilate then the first approach should be to eliminate those positions by a modification of the mining method. If this is not possible then a local increase in air velocity should be attempted. Much benefit can be gained from increasing low air velocities up to 1 m/s. Above 2 m/s, the cooling power of the air is little affected by further increases in velocity. In such cases the cooling power of the environment can be improved only by reducing the wet bulb temperature. Small compressed-air or electric airmovers may be used for local increases in air velocity. Partial recirculation of the air is another technique which may be considered provided that it is not prohibited by national mining law. In this case, the throughput of air must be such as to prevent a hazardous build up of toxic or flammable contaminants. Overheating from natural sources will not occur provided that the VRT is less than 35°C. It is also sometimes possible to inject fresh air into intermediate points along the working areas. This can be accomplished by a suitable layout of airways or, more locally, by air ducts.

If none of these techniques achieves the required environmental conditions then cooling plants must be installed. This can vary between local spot coolers and large-scale refrigeration plants. Heat transfer from the strata increases as air temperature decreases. Bulk air cooling is therefore inefficient. A more typical arrangement is illustrated in figure 13 with a refrigeration plant piping chilled water through a reticulation system with insulated cold pipes to heat exchangers sited in the working areas (7). The heat from the condenser water is dissipated either on the surface or through an underground cooling tower.

Underground air conditioning has long been practised in the deep metalliferous mines of the world. In South Africa underground refrigeration capacity continues to increase at a rate of approximately 10 per cent each year. However, the concentration of electrical machinery in European coal mines is also leading to the introduction of spot coolers. The German Federal Republic has already commenced the installation of refrigeration plants at collieries.

Perhaps the most important planning criteria are firstly, that the air should be cooled at or close to the working area and not at some remote point along the intake route, and secondly, that the system must be flexible. It is for this latter reason that chilled water reticulation systems can be so successful. The chilled water can be diverted to the area where cooling is most urgently required.

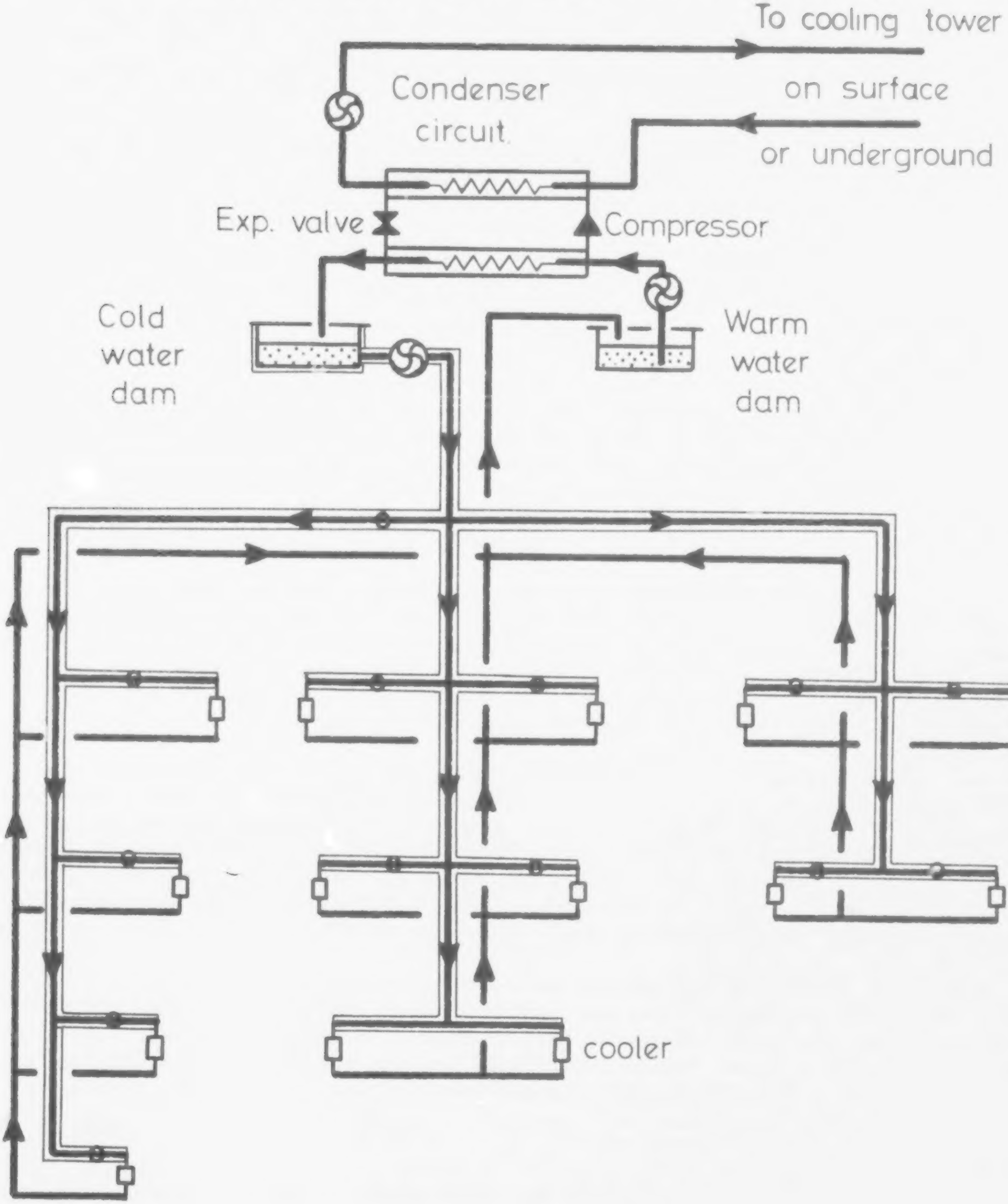
Experimental work is progressing in South Africa into the use of personal ice-cooled jackets. Other developments are leading to air-conditioned cabs on large cutting and/or loading machines.

Noise

Physiological effects of noise. The description of noise which is perhaps most popularly quoted is that it consists of unwanted sound. Together with other forms of environmental pollution, the problem of noise is attracting considerable attention. Noisy industrial environments lead not only to physical damage to personnel but also inhibit communication, cause frustration and may lead to mental unbalance. The repercussions on productivity and safety are obvious.

The physiological effect of noise differs considerably among individuals. The ear receives noise directly from the propagating source and also as reflected waves. The range of human hearing is enormous, the threshold of pain being at a

Figure 13. Mine refrigeration and water reticulation system


To cooling tower
on surface
or underground
Condenser
circuit
Exp. valve
Compressor
Cold
water
dam
Warm
water
dam
cooler

noise intensity some 10^{14} times greater than the threshold of hearing. This has led to the adoption of a logarithmic scale for the measurement of noise levels, the basic unit being the Bel. An increase of 3 decibels represents a doubling of noise intensity.

Working in a noisy environment has two distinct effects on the ear. Firstly, there is a temporary shift in the threshold of hearing which recovers progressively after the noise is removed. This may be of the order of 20 to 30 decibels for young people. Secondly, repeated and prolonged exposure to noise causes a cumulative permanent hearing disability. Contrary to age-induced hearing loss, the permanent threshold shift due to noise increases rapidly at first and then slows down.

The effect depends not only upon the noise intensity but also upon the exposure time. The concept of "equivalent continuous sound level" (L_{eq}) represents the continuous level of sound energy over an eight hour period which has the same effect on hearing as the varying noise levels actually received. Every attempt should be made to limit industrial environments to an L_{eq} of not more than 90 decibels (A)*. Beyond this limit, personal protection should be provided. Persons exposed to noisy environments should have regular audiometry checks in order to detect the onset of hearing loss.

Noise sources in mining and their control. Most industrial noise originates as machine vibration. Virtually every mining operation involves the propagation of sound. Not all of this should be regarded as undesirable noise. It is important that moving vehicles should be audible above the background noise level. The four most prolific sources of noise in mining are compressed air drills, auxiliary fans, compressors and mineral beneficiation plants.

The control of machine noise should commence at the design stage, the aim being to minimize vibration, impact and sudden gas expansion. In compressed air drills, some 87 per cent of the noise emanates from the exhaust, the remainder being caused by the impact between piston and drill steel (12). Noise levels in tunnelling operations with three drills may reach 120 decibels (A) with an L_{eq} of 114 decibels (A). The noise level can be reduced by about 6 decibels with exhaust silencers. Unfortunately, these also reduce the rate of drill penetration.

Auxiliary axial fans may produce an L_{eq} of over 90 decibels (A). This is caused mainly by the highly turbulent vortices shed from the fan blades and the rotation of the impeller. Fan noise is accentuated by higher pressures and increased air velocities. Sound-absorbing silencers in the air ducts can be fairly effective. Centrifugal fans produce considerably lower noise levels.

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* The (A) refers to the type of frequency weighting network used in the noise level meter. An A-weighted instrument simulates the human ear.

Multiple compressors within a confined space can give rise to L_{eq} values approaching 100 decibels (A). A common bedplate should be used for a motor/compressor unit. Properly designed resilient mountings will reduce conducted vibration and flexible couplings will prevent transmission along pipes. In underground locations, reflected sound can be reduced by lining the airway with acoustically absorbent material. Metallic sheet lining should be avoided.

Mineral preparation plants may produce L_{eq} values of well over 100 decibels (A). This is partially due to the increased exposure time of longer shifts. Wherever possible, machinery situated inside surface structures should not be bolted rigidly to the framework of the building. This avoids the structure itself becoming a radiating source. Most preparation plants have a number of machines in close proximity, many of which rely on vibration for their operation and are therefore intrinsically noisy. In such circumstances the majority of noise received by personnel is reflected sound, and there is little benefit in removing a single noisy machine.

All machinery becomes noisier with age - worn gears, bearings and loose plates can make a noisy machine quite intolerable. It is important that equipment should be correctly maintained.

After all practicable steps have been taken, there still remain numerous sites where unacceptable noise levels exist. If possible, the noise source and the operator should be separated. This can sometimes be accomplished quite cheaply by a re-arrangement of inlet and exhaust ducts or by simple remote instrumentation. In some situations, it may be necessary to place the machine operator in an enclosed compartment. To be effective, acoustic cabins must be designed and sited with care. The most important factors are the use of cavity walls, sound absorbent materials, isolation from external vibrations, the sealing of orifices surrounding pipes or cables and a wide field of view.

The final weapon against excessive noise is the use of personal protection. Temporary ear plugs of glass down (not cotton wool) are effective, cheap, hygenic and disposable. The more durable types of ear plug made from rubber, moulded acrylic or waxed fibre are also relatively inexpensive. It is important that they fit tightly and for this reason they may become uncomfortable when worn for long periods. They are also difficult to keep clean in the mining environment. The most effective personal protection is provided by ear muffs. These are rigid cups, filled with fluid or foam, which cover the entire ear. Here again, ear muffs may not be comfortable for more than a short period of time.

Mine lighting

The importance of illumination. Although the invention of the flame safety lamp in 1815 overcame the problem of illumination without explosion in coal mines, it was not until the introduction of the electric cap lamp in the 1920s that nystagmus was virtually eliminated. It is in recent years only that the scientific and engineering method has been brought to bear on mine lighting. The criteria are no longer those of eye diseases arising from inadequate illumination but the effect of lighting on safety, performance and morale.

As the mechanized equipment used in modern mining becomes more sophisticated it is increasingly important that the level of illumination should enable men to work safely in the vicinity of the machines. The performance of working men increases rapidly with illumination up to a certain level of lighting. Once this adequate level has been reached further illumination will have no beneficial effect on working performance.

The total range of brightness over which the human eye can perform is more than 10 million to one. However, the eye cannot respond to this full range at any one time. A large degree of "adaptation" may occur when moving from one location to another, the maximum level of adaptation being about 50 to 1. The accuracy of perception depends not only upon the level of illumination but also upon the angle subtended at the eye by the viewed object (measured in minutes of arc) and the contrast in brightness with the background.

Planning of mine lighting. For many years, it has been commonplace to use mains lighting for important underground locations such as shaft bottoms and main transport roadways. Unfortunately not all of these installations have been planned in a completely rational manner. It is important that the mine environmental engineer should be aware of the important criteria in designing a lighting system.

The prime consideration should be the ability to see and not necessarily the way in which a source illuminates. The vast majority of light which enters the eye is reflected light. Plans must therefore be drawn up with the emphasis placed on reflected light rather than on the light source itself. Provided that an adequate over-all brightness level is provided attention should be focused on luminance patterns (shades of contrast) to ensure efficient sight. The general illumination of task and background is often the same within the mining environment. Thus, suitable levels of contrast can be achieved only by choosing suitable reflection factors such as whitewashing of roadway surfaces or the use of luminescent paint on moving vehicles.

The level of illumination deemed adequate will be determined by the task, safety and the control of glare. The lighting in adjacent areas should be correlated to avoid sudden changes in brightness. Ideally, the brightness levels should change suddenly by a factor not over 3.5. In many illuminated places underground, mains lighting must necessarily be supplemented by the directional lighting of cap lamps for detailed perception. Although mains lighting at the places of mineral winning is not yet commonplace, it is nevertheless increasing with mechanized mining. The problems of safety and maintenance are being overcome by the use of intrinsically safe, low-power fluorescent tubes which may be fitted as integral parts of mechanized face equipment (2).

Mine lighting fittings should be chosen on the basis of cost, long life and low maintenance. They should be reasonably dustproof to remain efficient and may have to be flameproof with arrangements made for cooling. The light sources suitable for underground use are tungsten filament, tungsten halogen and low-pressure mercury vapour lamps. The latter source gives the familiar tubular fluorescent fitting used widely in permanent installations on mines today. It combines long life with a relatively large illuminating area of low brightness, thus reducing the probability of glare. The positions of lighting fittings should be considered carefully. To be most effective fluorescent tubes in mine airways should be fitted in line with the airway rather than across it.

Other types of lighting fitting available for mines include air turbine lamps for sites which do not have electricity and light sources included on moving machines. It is important that all light sources and relevant reflecting surfaces be maintained to ensure continued efficiency.

Despite the improvements which have been made in mine lighting installations, it is envisaged that the miner's cap lamp will remain his primary light source in the foreseeable future. Research work continues to seek methods of reducing the battery weight whilst maintaining an acceptable light output of 1.2 mean spherical candlepower after nine hours.

Radiation

The radiation hazard in mines. Owing to geochemical recycling, uranium and its daughter products are present in virtually all soils and rocks of the earth's crust. On a world-wide basis, the mean concentration of uranium in rocks is approximately four parts per million. Radon gas and alpha particles are constantly being emitted from all natural sources including building materials. The degree of radiation depends upon a number of factors including concentration of uranium. For this reason, particular precautions against radiation must be taken in uranium mines although radon gas may occur in other forms of mining (15).

The long-lived uranium 238 isotope decays down through the uranium series emitting alpha, beta and gamma radiation. At an intermediate stage radium 226 decays to radon 222. This is a chemically inert gas which has a half-life of 3.825 days and emits the doubly positively charged nuclei of helium atoms known as alpha particles. The radon 222 decays to the radon daughters including the alpha emitters radium A (half-life 3.05 min) and radium C' (half-life 164 micro sec). These are solids most of which adhere to the surfaces of dust, smoke and water particles.

Ionizing radiations can cause both somatic and genetic effects on the human body. Alpha radiation gives rise to intense ionization over short distances. Fortunately, alpha particles have little power of penetration and are almost entirely absorbed by the body skin. There is therefore little danger of external radiation damage. It is when radon gas or radioactive dust is introduced into the body, either by respiration or ingestion that internal radiation damage may occur. The bombardment of lung tissue and respiratory tracts by alpha particles may lead to cumulative lung damage and cancer. Ingested alpha emitters tend to become fixed in the bones and may result in blood disorders with possible eventual damage to the bone structure.

Radioactive isotopes are beginning to be employed in the horizon control of power loading machines in coal mines. Provided that adequate shielding precautions are taken these offer no radiation hazard.

Control of radiation. The mechanism of radon emanation from rock is complex. The current theory is that recoil of the radon atoms occurs within the crystalline matrix leading to a relatively rapid diffusion along planes of weakness and cementing material to the particle surface. With certain exceptions, the rate of radon release is proportional to 1/d where d is the rock particle diameter (17). The radon is released mainly from the outer shell of the rock particle at a rate depending upon the concentration of uranium, the permeability and porosity of the rock and also the radon concentration gradients which are set up within the rock and in the air or water surrounding the particle.

The practical implications of this mechanism of radon emanation are that there should be as little comminution of uranium ore as is practicable in the mine and that broken ore should be cleared from the mine with minimum delay. Here again, control of the mine environment commences at an early stage of mine planning.

In established mine workings the most immediate control of radon concentration is effected by dilution. Fresh-air splits should be provided to each working area and series ventilation kept to a minimum. In some cases this will involve additional airway capacity and enhanced fan duties together with improved auxiliary ventilation and perhaps air heating. Recirculation with dust filters is of limited value - the radon daughters deposited on dust may be removed but not the radon gas which will continue to replenish them.

The air quantities required may be determined from local experience on the basis of mine production. This will vary according to mining practice but a value of 0.05 m^3/s (100 ft^3/min) per ton of daily rock broken is an acceptable guide (15). Research work in Canada has led to the establishment of procedures for the forecasting of air quantities required (16).

In addition to the factors already outlined, it has been found that radon emanation is also effected by barometric pressure, temperature, geologic anomalies and ventilating air velocity. Radon concentrations do not, therefore, remain constant in a mine. Radioactivity should be monitored in uranium mines and included in surveys at least every 4 months.

B. Ventilation planning

The role of planning

As underground mines become deeper and workings more extensive, the costs of ventilation increase. It becomes increasingly important that the power expended in creating air movement and providing acceptable temperatures is utilized efficiently. All too often in the past, there has been little attempt to plan the ventilation in a rational manner. In such mines workings have proceeded until airflows become inadequate. The short-term solution would then involve the installation of additional underground fans to boost the airflow in one section of the mine, often at the expense of others. The result of such procedures is the development of an expensive and inefficient ventilation system. This can lead to premature cessation of mining operations or necessitate a costly scheme of re-development. The two main reasons for inadequate ventilation planning have been the uncertainty of mine layout due to varying mineral grades and the complexity of calculations involved in the forecast of airflow patterns.

In modern mining, economics demand that ventilation be an integral part of underground mine planning. Ventilation and air-conditioning costs may otherwise become prohibitive. Computerized techniques have been developed which enable the environmental engineer to forecast the positions and duties of the fans and cooling plant.

The quantities of air required in the working areas are governed by difficulties arising from gases, dust, temperatures, air velocity or the removal of fumes. Unfortunately these do not, by themselves, determine the pressures and airflows to be provided by the fans - nor do they specify the best position for each fan. Such factors are also affected by the position, size, length and condition of each airway in the system, including all leakage paths together with the variations in natural ventilating effects. It follows, therefore, that if accurate and reliable ventilation plans are to be drawn up then airflow patterns and corresponding fan duties must be computed for a range of possible mine developments (1, 18, 19).

The organization of ventilation planning procedures

Since the mid-1960s, the techniques of ventilation planning have been revolutionized by the use of digital computers. Manual calculation methods are suitable for simple localized airway systems only and analog computers are relatively slow, inflexible and limited in capacity. To make the best use of efficient, fast and precise techniques of calculation, it is necessary to obtain correspondingly accurate and detailed data on which to base the analyses.

The sources of basic information required for airflow planning are ventilation surveys. Airflows are normally measured by anemometer traverses whilst the remaining parameters of frictional pressure drop, energy dissipated, airway resistance etc., are obtained from observations taken on hygrometers, barometers and manometers (29). In order that objective forecasts of ventilation requirements can be made for the further developments of an existing mine, it is essential that such surveys be carried out. Furthermore, the surveys themselves should be planned most carefully by environmental staff in order to ensure that all relevant information is obtained and that instruments are maintained within known calibration.

Little manual calculation is necessary for the ventilation planning of a modern mining unit. Figure 14 shows the flow of information through the airflow planning procedures. Observations made during surveys are processed by computer. This enables current plans to be compiled, showing the present state of the airflow system. The survey programmes also produce data which can be used as a basis for

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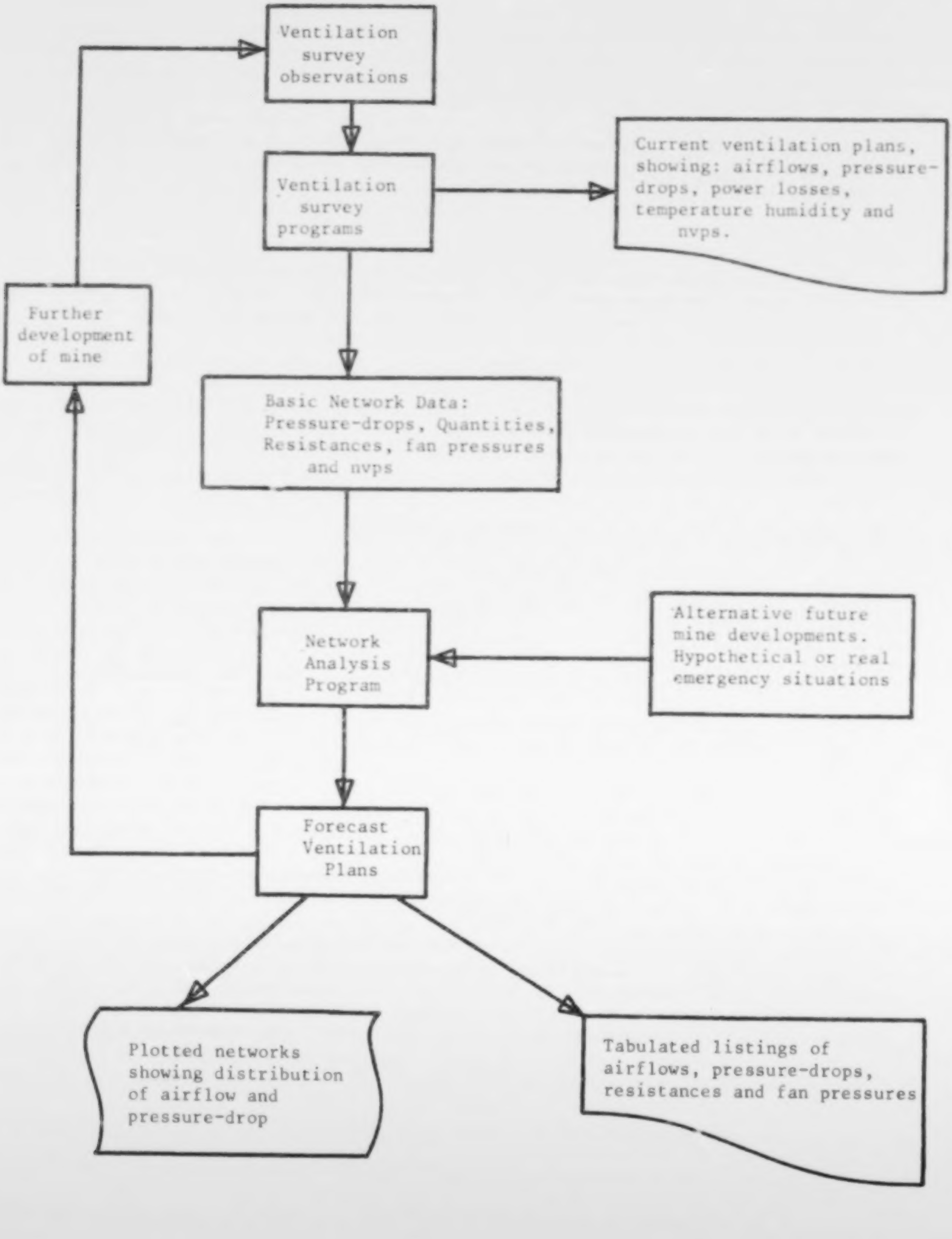
Figure 14. Information flow chart for ventilation planning


Ventilation survey observations
Ventilation survey programs
Current ventilation plans, showing: airflows, pressure-drops, power losses, temperature humidity and nvps.
Further development of mine
Basic Network Data: Pressure-drops, Quantities, Resistances, fan pressures and nvps
Network Analysis Program
Alternative future mine developments. Hypothetical or real emergency situations
Forecast Ventilation Plans
Plotted networks showing distribution of airflow and pressure-drop
Tabulated listings of airflows, pressure-drops, resistances and fan pressures

ventilation network analyses. The network programmes enable the environmental engineer to investigate the airflow distributions which would result from any development of the mine or in the event of a given emergency condition. Furthermore the duty and distribution of fans can be investigated for future extensions to the mine.

All underground mines will have a skeleton layout of semi-permanent airways that changes slowly. This basic network may be retained on magnetic store at the computer centre and updated as new surveys become necessary. The environmental engineer may then rapidly evaluate the effect of any proposed change to the system. If the mine has a direct access link to the computer then modified airflow systems can be calculated within minutes. This facility provides a powerful tool - not only for routine planning but particularly in an emergency such as an underground fire or explosion.

A number of mine ventilation network programmes have been developed and become a routine part of ventilation planning procedure (18, 19). In the United Kingdom, the National Coal Board computer system analyses more than 3,000 ventilation network exercises per year. More sophisticated network programmes are now beginning to appear that analyse detailed airflow patterns within the stopes for given mining methods (20).

The first approach to problems of gas, temperature, humidity and to a lesser extent dust, is to increase air quantities. However, with modern mining methods and rates of mineral production, this may lead to unacceptable fan power costs and intolerable air velocities. In some cases (and where permitted by law), the efficiency of the airflow system can be improved by controlled recirculation. Gas drainage techniques may sometimes be utilized to advantage in controlling pollution of the ventilating airstreams.

In the past, the planning of airflows to deal with problems of heat and humidity has largely been based on empirical data (9). This has led to general criteria such as cubic metres per second of air per ton of mine production. More precise predictions have been difficult owing to the interaction of a large number of variables including mine layout, method and rate of production, type of support, age, length and wetness of airways, clearance of broken mineral, condition of intake air, VRT, and thermal conductivity of the native rock. Computer simulation techniques have been devised (10) that take all of the relevant factors into account and forecast the wet and dry bulb temperatures which will exist in planned mine workings. Such forecasts have shown good agreement with actual measurements taken underground (8). In planning for hot and humid environments, the main procedures are as follows:

(a) Define the environmental conditions that are acceptable. A value commonly chosen in South Africa for planning purposes is 31°C wet bulb. This may, however, vary according to the mining tradition of a country and the amount of manual work involved;

(b) Calculate the air quantities required, using the minimum possible refrigeration;

(c) Carry out a ventilation network analysis and determine the fan duties required;

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(d) Repeat the computations with reducing air quantities and increasing increments of refrigeration.

A cost optimization may then be carried out.

C. Beyond the mine precinct

The balance of industrial growth and ecology

The alarming trends of land, water and air pollution in recent years have resulted in controversy surrounding the control of further technical innovations. The experts differ in their opinions from those who see the quality of human life severely repressed by the end of the century, to those who are optimistic that technology will itself find the solution to problems of pollution. National and international conferences have been successful in generating public awareness and in prompting governments and industries to take appropriate action.

Virtually all industrial activities cause some pollution. The time has passed when the self-correcting powers of the atmosphere, waters and land can be assumed to be capable of handling mankind's waste. One method of maintaining a clean environment would be to impose severe restrictions upon all industrial development. The result would be disruption of national economies, a fall in international trade and severe drops in living standards. Clearly, a balance must be sought between industrial growth and ecology. In the first instance it is the responsibility of governments to set realistic environmental standards, preferably by international agreement, to which all industrial activities can, and must, conform. It is then the task of industry to devise means of remaining viable whilst meeting those standards.

The maintenance and continued evolution of industrial societies are totally dependent upon the mineral resources of the earth's crust. The mining industries of the world are basic to all others. It follows, therefore, that the cost of maintaining acceptable environmental standards in and around mines must be met. Quality of the mining environment can no longer be regarded as peripheral but must be accepted as a major factor and a real cost in the planning of mining operations.

The major forms of pollution caused by mining are despoliation of the land and disturbance of natural chemical cycles by adulterated drainage waters (22, 23). Both of these have resulted in clashes between conservationist groups and mining exploration. The arguments are sometimes more emotive than rational. In general, mining companies are accepting their responsibilities, often going far beyond legal requirements.

The remoteness of some mining locations is no reason for loose environmental control. In the colder climates, chemical and bacterial changes occur very slowly and a delicate ecological balance can be easily and irreversibly upset. With modern ease of travel, some mining locations in aesthetically attractive areas are becoming centres of tourism. This should not be discouraged by unreclaimed land or polluted waters. Mining communities themselves are no longer content to live in the dismal surroundings which have been so often the traditional feature of mining areas - and which have exacerbated the movement of population from the region.

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Despoliation of the land

Stabilization of tips. It is normal practice in underground mining to ensure that as much waste materials as possible remains underground. Nevertheless, it is inevitable that a great deal of tailings material and rock debris must be disposed of on the surface. The resulting tips, tailings ponds and spoil heaps should be constructed and treated in such a way as to render them safe from slip or erosion.

With modern rates of mine production it is essential that waste disposal is carefully pre-planned. Physical and vegetative methods are the most powerful means of tip stabilization. Purely chemical methods involve spraying the area with compounds designed to create an air- and water-resistant crust. Physical stabilization to prevent excessive dust from dried tailings areas includes covering with slag, harrowing in binders such as straw or wood bark and the use of windbreaks.

The need to ensure the stability of mine tips was never more dramatically emphasized than at Aberfan, Wales, when on the morning of 21 October 1966 a water-saturated colliery tip moved down a mountainside to engulf the village school. One hundred and forty-four men, women and children died. The Mines and Quarries (Tips) Act which followed in 1969 defines the precautions that are necessary to ensure the stability of tips and tailings areas (21). The physical properties of the tipped material must be known. New colliery tips in the United Kingdom must be based on good site preparation. The tips must be stratified with provision made for the prevention of a build-up in internal water pressure. Adequate finger and blanket drains must be constructed into the tip and surfaces consolidated to assist surface water run-off. Particular precautions must be taken for tips erected on sloping ground and for the banks of tailings ponds. Research continues into the time dependence of tips, flow slides, stratification, consolidation, drainage and subsidence.

Reclamation. Rehabilitation of mining waste areas is becoming established practice in a number of countries. By far the most successful technique is vegetative stabilization, with much of the pioneering work having been carried out on the North American Continent. Vegetation eliminates dust from tailings areas. This improves not only local living conditions but also eases the problems of machine maintenance in the area. A vegetative covering regulates the effect of heavy rainfall, reducing sliding and erosion (27, 28).

Strip mining wastelands can often be reclaimed by smoothing out the land contours and employing conventional agricultural methods. Tailings are more difficult to reclaim, varying from the highly acidic to very alkaline. Erosion and the local climate may have caused gullies and hard crusts on the tips. Untreated tailings remain a sterile eyesore, a source of air and water pollution and a potential danger to the community.

Mill wastes will typically consist of sands, excessive salts and toxicants.

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The "soils" necessary for the promotion of vegetation contain minerals, water, nutrients and micro-organisms. The latter two are missing from tailings and must be supplied by a correct choice of fertilizers, additives and bacterial growth promoters. The correction of pH may be achieved by the addition of ground limestone or basic tailings if acid conditions prevail, and sulphuric acid or acid tailings to counteract alkaline material. Flooding has been found to be ineffective in leaching out acidity although some success has been achieved using fine water sprays (28).

The procedure of vegetative reclamation commences with widespread sampling of the tailings area to determine variations in pH and size distribution. This enables an initial choice to be made on additives and fertilizers. Tests may then commence on "soil"/plant/climate compatability. Contouring of the area should be carried out by track or tyre-mounted vehicles suitable for the tip material. In general, soft rounded contours give a pleasing appearance. Localized depressions lead to water traps and should be avoided. The additives and fertilizers are then introduced to correct pH and provide the nitrogen-phosphorus-potassium-bacteria basis required by plant life.

The plants must be introduced in a carefully planned order. Initial planting of trees is not likely to be successful. It is necessary to establish a vegetation cover first of all by grasses and the nitrogen fixing leguminous plants (Tre-foils, clovers). A good grass/legume mixture can often be established rapidly and begin the process of surface stabilization and soil promotion. The plants chosen should be self-supporting in the prevailing climate. The importation of soil should not be undertaken on any large scale but may be necessary to combat heat reflection where light-coloured tailings face major solar radiation.

The seeding should be followed by mulching - the introduction of fibrous organic material to conserve water, and reduce wind erosion during germination. Sand blasting will eliminate young seedlings most effectively. Emulsions may also be employed to stabilize the surface until the network of plant roots is established.

The planning of land reclamation should commence before mineral winning begins. The extraction and reclamation processes should be integrated. Ways should be sought of putting the excavations and structures of mining to a useful purpose. Surface mining can improve drainage and offer flood protection. Reclaimed land may be used for agriculture, fisheries, recreation or urban development. Flat tailings areas can be at a premium in rugged country for parks or airfields. Similarly, underground excavations may provide valuable storage for water, oil or gas. In these ways, the management of the ecosystem and the exploitation of minerals should be complementary. The mining engineer should consider means of converting tailing ponds and spoil heaps from liabilities into assets.

Water pollution

It has been estimated that 97.5 per cent of the earth's water is contained in the seas and oceans. Of the 2.5 per cent fresh water some 2.2 per cent is held in the polar ice caps leaving a mere 0.3 per cent contained in land drainage systems. This, however, represents more than one million cubic miles and is a vital factor in the ecological balance (26). The great majority of chemical

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elements undergo a geochemical cycle which involves rocks, soils, water, air and biological processes. When an element or associated compound becomes out of balance at any stage then it is possible that the complete cycle may be disturbed. This is most likely to occur by water pollution, either directly or by fall-out from airborne substances. Fortunately, the soils and sediments of drainage areas have considerable sequestering powers enabling most naturally occurring deleterious substances to be rendered harmless. It is only when the concentrations of toxic substances exceed the sequestering ability of the environment that serious problems arise (22).

The most dangerous substances produced by mining activities include sulphuric acid, copper, selenium, tellurium, fluorine, nickel and radioactive elements. The reclamation of a polluted drainage system is a long and difficult business. The emphasis must be placed on prevention. Lakes are more vulnerable than rivers because of the greatly increased residual time during which chemical and biological disturbances can occur. In the past, it has been the aim of mining concerns to remove all solids from water leaving the area. Whilst this is still necessary it can no longer be assumed to be sufficient. Toxic substances dissolved in clear water may continue to accumulate or initiate ecological disturbance at lower points in the drainage system, including the sea itself.

In many cases, mines can hold pollution within the mine perimeter only by large-scale water recirculation systems - particularly in areas which are naturally short of water. Because of the large storage, settlement and conditioning areas required major earthworks may be necessary (24). Furthermore, the increased concentrations in the recirculating water may necessitate improved water conditioning plant. A typical treatment would be the addition of fine limestone pulp to acidic waters resulting from the treatment of sulphide ores (25). Particular precautions are necessary to reduce the radium content of waters from uranium mines. Some 99.75 per cent of the radium in the tailings from the uranium mines of Ontario are held within the solids and it is considered most unlikely that any further significant leaching will take place from the dried sediment. However, in order to prevent a radioactive build-up downstream, it has been found necessary to lower the 0.25 per cent dissolved radium in the overflow water by a barium chloride treatment. Additional precautions are taken against accidental overflow of solids and downstream monitoring is carried out (15).

Air pollution

During the London smog of December, 1952, some 4,000 people died within a period of 10 days. Incidents such as this have resulted in widespread public concern about airborne toxicants. A number of countries have introduced Clean Air Acts in order to control the concentrations of sulphur dioxide, nitrogen oxides, carbon monoxide, particulates and photochemical substances in the atmosphere.

It is the uses to which mine products are put rather than the mines themselves which result in atmospheric pollution - in particular the combustion of fuels for heat and energy. However, mining companies should ensure that the exhausts from furnaces are treated by chemical, filtration or precipitation methods to reduce unacceptable concentrations of toxicants. Similarly, any process involving large throughputs of air, such as ore drying, should ensure the efficient removal of particulates before exhausting to the free atmosphere.

Dust problems from surface plant may be reduced by appropriate shrouding and dust-extraction units. This is particularly important on conveyors and at dry ore discharge points. The amount of dust raised by moving vehicles in open pits may be controlled by water sprays and roadway consolidants.

D. Conclusion

This paper has reviewed a number of environmental hazards which arise in the practice of mining engineering, together with some of the steps which may be taken to control them. Many of these problems are exacerbated by modern techniques of mineral winning and processing. The quality of the environment in and around mines has all too often been regarded as of minor importance compared with the mining processes themselves. In many cases, the roles of ventilating, air conditioning and water treatment plant together with land reclamation have been considered only after the major mining and processing methods have been decided. Such an attitude leads to high costs in the long term, undue human suffering and even premature closure of mines. Environmental control must be considered as an integral part of mine planning.

Ventilation in mines is sometimes regarded as a sword of Damocles suspended by a fine thread above the head of the mining engineer. Provided all goes well then he tends to forget about it. If it fails, the consequences can be disastrous. The same is true of other forms of environmental control. Some of the economic advantages inherent in a well-planned industrial environment are indirect and often long term. On the other hand, the cost of neglecting them may be catastrophic.

The mining industries of the world are vital to industrialized societies. Equally important is that the planet earth should remain a fit place in which to live. Mining engineers must accept a responsibility to ensure that the two remain compatible.

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IV. ROCK MECHANICS AND GROUND CONTROL

D. G. F. Hedley and G. Herget*

Rock mechanics is the branch of science which deals with the stresses and displacements in rock and their effect on stability of rock structures. After excavation, either on surface or underground, the load, originally borne by the excavated rock, must be borne by the surrounding rock. Depending on the strength of the rock, this increase in stress can cause failure. Many factors affect the forces acting on rock and the strength of rock and it is not an easy matter to determine stability. However, the purpose of rock mechanics studies is to obtain some degree of predictability on how rock behaves around mine openings.

Rock mechanics is a relatively young science and has not reached the stage of completely designing mining excavations. At present, research provides relevant information to mine operators which assists them in making decisions concerning mining layouts, sequence of mining, level of extraction and artificial support systems. The lack of a scientific basis has not hindered the mining industry in the past. Over the last few decades the major trends in mining have been towards increased mechanization and larger equipment for drilling, raise boring, transportation and automated support. The design and layout of mine openings, both underground and on surface, are still based mainly on experience and trial-and-error. However, the picture is changing in that the vast low-grade deposits mined by open pits require careful slope design and prediction of ground conditions; otherwise, they are not economical. In underground mines, the trend is towards concentrated production from a few highly mechanized stopes. Ground conditions and stability in these stopes then assume a much higher level of importance.

There is an increasing awareness by the mining industry of the need for more information on ground conditions and how they influence mining. This is reflected by the number of groups of engineers and scientists presently engaged in rock mechanics research. In Canada, there are four distinct groups covering the spectrum from practical to basic research and from short- to long-term objectives. As a broad guideline, the areas of research of each group can be classified as follows:

(a) Large mining companies usually employ their own rock mechanics engineers. Their work is concerned mainly with mining problems which require immediate practical solutions (within weeks). Existing knowledge and experience are used to deal with these problems. Relatively simple measurements are taken and analysed on a cause-and-effect basis;

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(b) Mining consultants are usually involved with problems which require solution within a few months. Again the work is practically orientated. More sophisticated methods of testing and analysis are sometimes used;

(c) Government research establishments such as the Mining Research Centre in Canada are generally concerned with medium-term (1 to 5 years) research. The problems are more related to a section of the industry than to a specific mine. Work is done on instrument development, testing techniques, and adaptation of research from other engineering disciplines;

(d) University research departments tend to work on long-term (over 5 years) basic research in such areas as theory of failure, mathematical simulation, and laboratory testing.

This structure is not rigid; there are instances of practical research being done by universities and sophisticated research by mining companies.

This paper will describe certain factors that influence the stabilities of mine structures and excavations. Methods of measuring and estimating rock mass strength and the stresses acting on rock are discussed, and the use of rock mechanics investigations is illustrated by several practical applications.

A. Strength of the rock mass

Strength is the stress a material can withstand up to its failure. Rock masses consist of a large variety of materials which may be weathered, decomposed or fresh. During geological history, tectonic deformations have fractured and weakened the rock materials; therefore, rock masses display great variations in strength. It is important to distinguish between the continuous rock material and the zones of weakness of fractures (discontinuities, figure 15).

This subdivision is a convenient one to use in any investigation. However, when it comes to the analysis of rock mass behaviour, both units interact to such a degree that even if both units are well known, the rock mass behaviour is often difficult to predict.

Strength and deformation of rock materials

If a rock is subjected to stress, the resultant deformation or strain can be plotted as shown in figure 16 (1, 2).* For any type of rock there are five typical zones:

(a) Zone of closure of existing open cracks;

(b) Zone of linear elastic deformation (without crack development);

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* Numbers in parenthesis refer to references listed at end of report.

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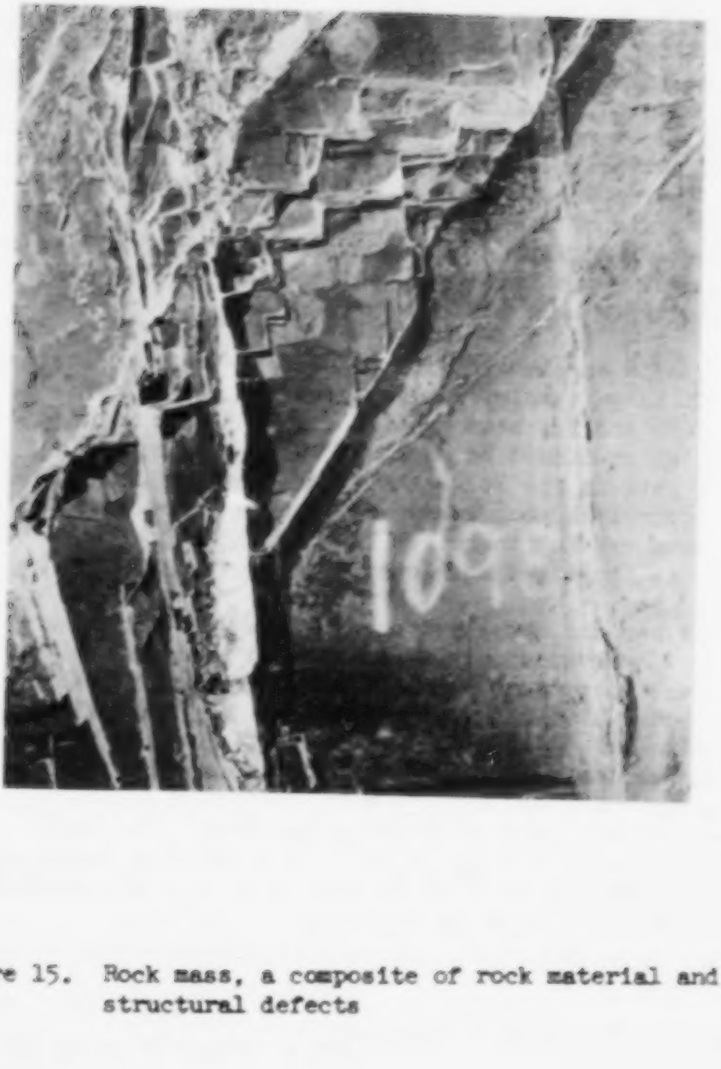

Figure 15. Rock mass, a composite of rock material and structural defects

(c) Zone of stable fracture propagation (deformation with development of fractures which stop propagating if load is not increased);

(d) Zone of unstable fracture propagation and failure (deformation with development of fractures which will continue to grow even if load is not increased);

(e) Zone of post-failure behaviour.

The extent of these zones will differ according to the rock type. The curve displayed in figure 16 can be obtained by loading a trimmed core sample, fitted with a suitable strain- or deformation-measuring device in uniaxial compression. Certain preparation requirements have to be observed because moisture, temperature, and end conditions influence the strength (3). Apart from various loading conditions (uniaxial, biaxial, triaxial), the volocity of load application and the duration of load will change the curve. For hard rocks, however, the variations must be orders of magnitude higher to achieve considerable changes in the stress-strain curve. To characterize the deformation behaviour of the rock on the basis of the curve in figure 16, we can calculate the secant modulus (deformation modulus = stress at failure/total strain up to failure) or the elastic modulus which is the tangent modulus for the centre straight part (zone of elasticity).

The volume of a sample will influence the strength obtained for any rock material in that large samples support lower stresses. This is well proven empirically (4, 5) and has been explained by the fact that all rock contains cracks or flaws which cannot be seen with the unassisted eye. For failure to occur along any path in the rock, the length and the orientation of the cracks to the maximum principal compressive stress are important. A larger volume of rock contains more cracks, therefore the probability of encountering critically-oriented cracks is increased and lower failure strength can be expected.

Stress analyses and observations of loaded specimens have shown that cracks developed during the loading stages tend to be parallel to the maximum compressive stress direction, and only after a hard rock sample is pervaded completely by vertical cracks will a collapse due to buckling and shear of the intact portions occur (6, 7). The strength of a rock will depend on its confinement and various theories have been proposed to correlate the maximum principal compressive stress (σ_1) and the confining stress (σ_3). The best-supported theories are those of Griffith and Mohr. The basis of the Mohr's strength theory is that failure occurs if the internal angle of friction is overcome on incipient surfaces of failure in either cohesionless soil or solid material. Coulomb's friction law states:

$$S = C + \sigma_n \tan \phi,$$

where: $\tan \phi$ = coefficient of friction,

σ_n = normal stress,

C = cohesion,

S = shear resistance.

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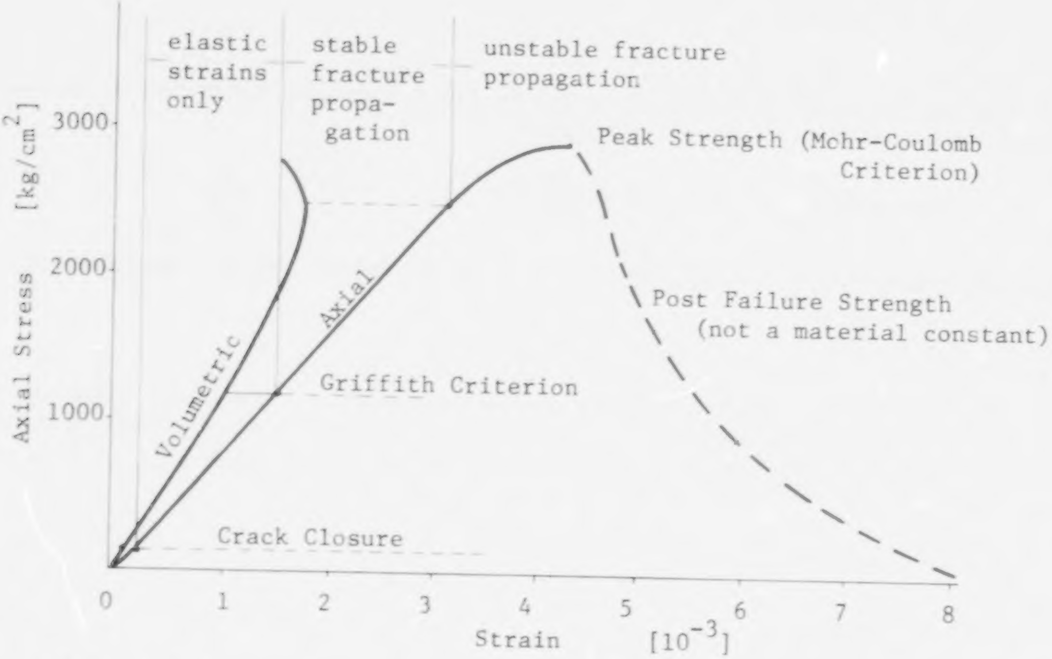

Figure 16. Stress-Strain Curve for Quartzite (Bienawski, 1969)

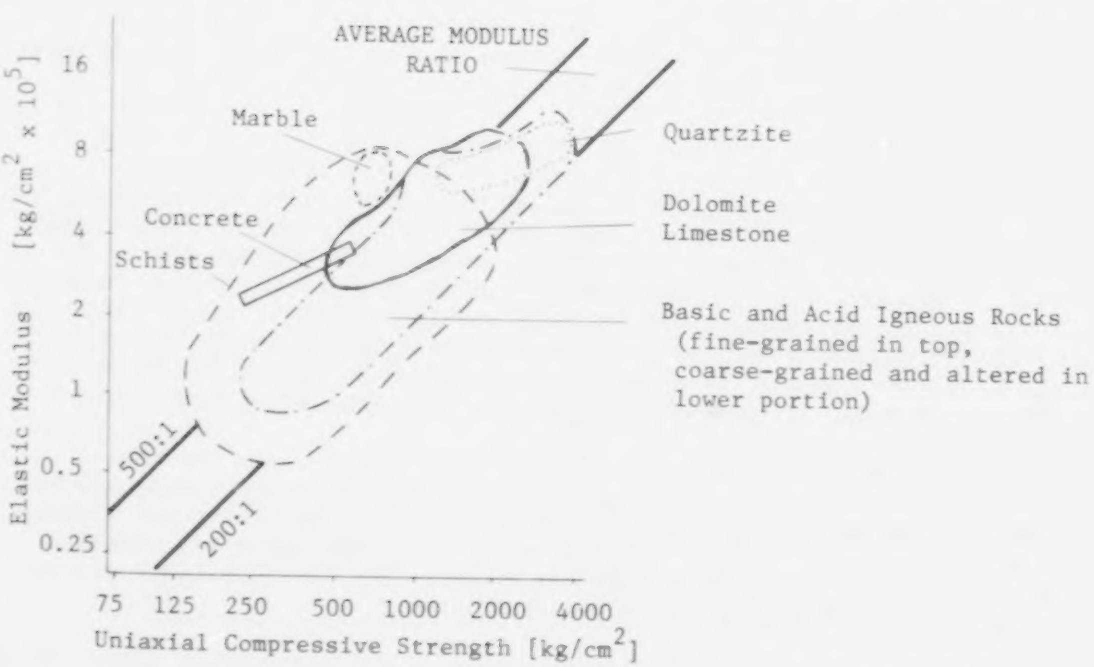

Figure 17. Distribution of Elastic Modulus and Uniaxial Compressive Strength (D.U. Deere and R.P. Miller, 1966)

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Calculating the shear and normal stresses for a specimen subjected to a biaxial stress system with $\sigma_1 > \sigma_3$, we obtain:

$$\sigma_1 = 2C \frac{\cos \phi}{1 - \sin \phi} + \sigma_3 \frac{1 + \sin \phi}{1 - \sin \phi}$$

The angle under which the failure surface is orientated to the maximum stress is:

$$\beta = 45 - \frac{\phi}{2}\ .$$

The Griffith's theory is based on the existence of cracks or flaws in rock and other brittle materials and determines the growth of these flaws under tensile stresses developed at the crack boundaries during compression.

$$T = \frac{(\sigma_1 - \sigma_3)^2}{8\ (\sigma_1 + \sigma_3)}$$

T = tensile strength

$$\text{when } \sigma_3 = 0, \quad \sigma_1 = 8T.$$

The predictions of this theory are not borne out in practice and, though the basic concept of cracks existing in brittle materials is sound, considerable modifications have been proposed which are beyond the scope of this paper. Failure theories are useful if a deeper understanding of rock behaviour is required; however, they will not relieve us from the obligation of testing rock materials if we want to use them in any type of structure.

A compilation of rock strength and deformation values is presented in figure 17. In many cases, it is sufficient to define the strength and deformation behaviour in approximate terms and only index tests are required. One testing unit, the Schmidt hammer, uses the rebound of a spring-loaded hammer held against the rock to estimate the uniaxial compressive strength (8). The other type of test is the point load test (9) which squeezes two points or steel balls into the opposite sides of a drill core specimen until it fails. With the help of calibration charts the uniaxial compressive strength can be determined. Another approach used mainly for softer rocks is given in figure 18, where consistency or hardness is used to determine the strength of the rock material (10)

Large variations in strength are observed even when testing the same geological formation, so, very often, just quoting a mean value is meaningless. If samples of a geological formation be grouped as sound, defective and broken, a normal distribution is obtained for each group for which means and standard deviation can be calculated; these data are valuable for design because we can assess probabilities of failure for given conditions.

After failure, the strength of a rock sample is not necessarily zero. It can be completely destroyed if the energy in the loading system is great enough and if the reaction of the system is sufficiently quick. This is normally the case

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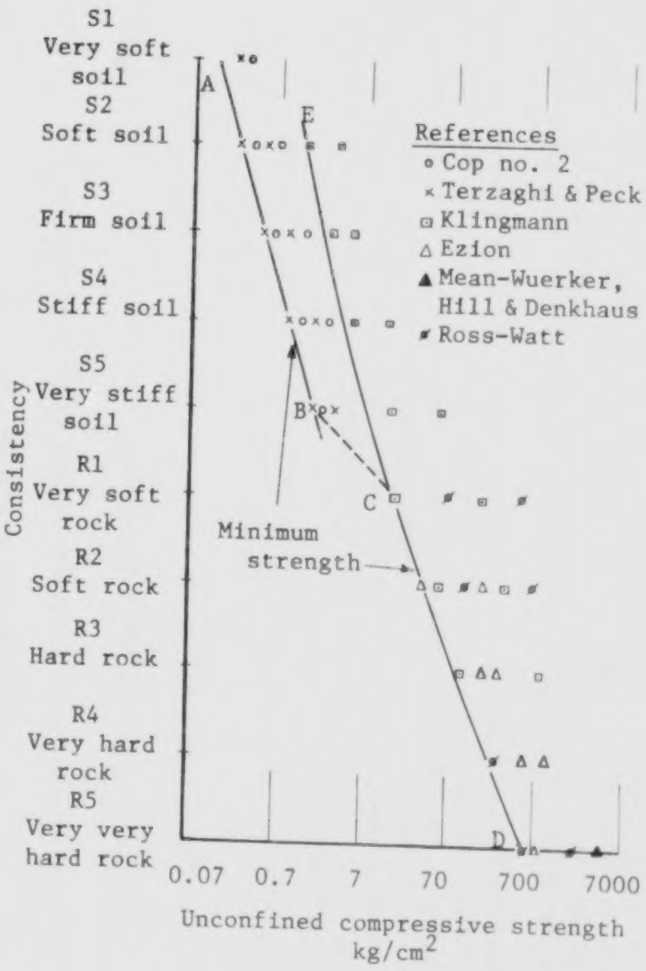

Description of Simple Consistency Tests	Minimum compressive strength [kg/cm²]	Cohesive strength [kg/cm²]
S1 (Very soft soil) - Easily moulded by hand, shows distinct heel marks.	0.14(-)*	0.02
S2 (Soft soil) - Moulded by strong hand pressure, shows faint heel marks.	0.21(1.5)*	0.04
S3 (Firm soil) - Moulded with difficulty, difficult to cut with hand spade.	0.42(2.2)*	
S4 (Stiff soil) - Cannot be moulded by hand, cannot be cut with a hand spade and requires hand-picking for excavation.	0.90(4.0)*	0.20
S5 (Very stiff soil) - Difficult to move with hand pick, requires pneumatic shovel for excavation.	1.75(7)*	0.75
R1 (Very soft rock) - Crumbles under firm blows with point of geological pick, can be peeled by a pocket knife.	17.5	1.7
R2 (Soft rock) - Can be peeled by a pocket knife with difficulty, shallow indentation made by firm blow of geological pick.	35.	5.5
R3 (Hard rock) - Cannot be scraped or peeled with a pocket knife, specimen can be fractured with single firm blow of hammer end of geological pick.	100.	17.0
R4 (Very hard rock) - Specimen requires more than one blow with hammer end of geological pick to fracture it.	218.	56.0
R5 (Extremely hard rock) - Specimen requires many blows with hammer end of geological pick to fracture it.	700.	100.0

Figure 18. Relationship between Consistency and Unconfirmed Compressive Strength A,B,C,D (Robertson 1970), *E,C,D (Robertson and Jennings 1968)

/...

in conventional ("soft") testing machines. A similar loading condition exists for planar failures in open pits (when the peak strength is overcome, the slide accelerates).

If a loading system is stiffer than the reaction of the failed rock specimen, the specimen can still support a considerable load. In many underground mining situations, pillars have been severely fractured but the distribution of load into surrounding areas has left the pillar in a state of operational safety but still partially effective as a structure. This will be discussed later in the description of the yield-pillar mining method.

Influence of geological fractures on the strength of rock

All rocks at the surface and at depth have been subjected in the geological past to processes of deformation up to the point of failure; therefore we observe fractures and weakness planes such as faults, shear zones, and joints (figure 15). Rocks with these defects are naturally weaker and excavations become less stable if these zones of weakness are critically orientated. Two problems arise in handling defective and broken rock:

(a) Detection of the location, orientation, extent and frequency of structural discontinuities, and their physical properties;

(b) Assessment of the effect of these structural discontinuities on the excavation.

Item one is obtained by engineering geological mapping of rock exposures and drill cores (11, 12, 13, 14). The most consistent way of mapping is achieved by using sheets which contain in short form all the questions which should be answered in the field. Handling of the information can be done by graphical analysis or with computer processing which can go to any desired complexity. The important goal in all cases is to get an accurate description of the rock mass in regard to rock types and defects; this can be obtained from maps of the area such as figure 19 and from models made up from perspex or glass sheets.

After the structural defects are known, their physical properties require investigation. Tests are therefore performed to determine the reaction of the discontinuities to normal and shear forces. This is often done by taking two blocks of rock which contain the fracture, clamping the lower block, applying a normal stress across the fracture and forcing the upper block to move along the fracture plane. The results are often expressed by the previously mentioned Coulomb friction law,

$$\tau = C + \tan\phi\,\sigma_n$$

With the fault submerged in water the relationship changes to

$$\tau = C + \tan\phi(\sigma_n - u),$$

where u represents the water pressure at the fault position, and

$$\sigma_n - u = \text{effective stress}$$

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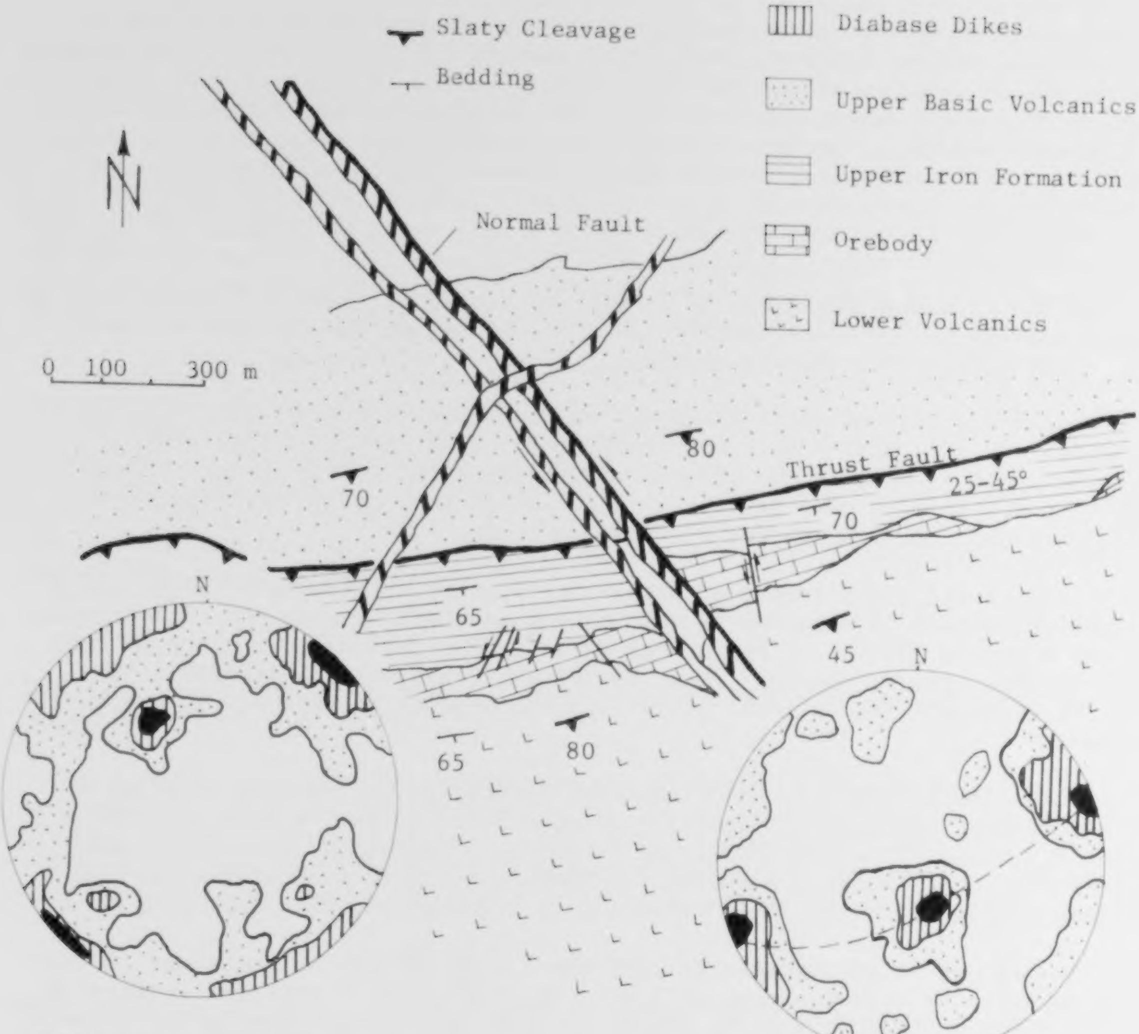
Slaty Cleavage
Bedding
Diabase Dikes
Upper Basic Volcanics
Upper Iron Formation
Orebody
Lower Volcanics
N
Normal Fault
0 100 300 m
80
70
Thrust Fault
25-45°
70
65
45
65
80
N
N

486 Shear Fractures
Contours at 1%, 2%, 3%
per 1% area
Maxima at N60° E/40° SE
and N130° E/90°
in lower hemisphere.

432 Extension Fractures
Contours at 1%, 3%, 7%
per 1% area
Maxima at N70° E/20° NE
and N160° E/90°
in lower hemisphere.

Figure 19. Example for Describing Geological Setting of a Mine Area

When testing geological discontinuities in the laboratory and in the field one should select testing loads relevant to the design objective. For smooth surfaces in rock, tan ϕ (coefficient of friction) varies between 0.5 to 0.7 (15). The cohesion value C is seldom constant; it shows large variations with normal stress and surface roughness. Therefore it is usually sufficient, for the given test conditions, to express the shear resistance developed along a fracture by,

$$\tau = \sigma_n \tan \phi .$$

The deformational behaviour of a fracture can be determined by monitoring the sliding and the stable blocks during the shear tests and one can observe large differences between the failure strains of broken rock and sound rock.

Knowing the rock material strength, the location, and physical properties of geological weakness planes does not immediately reveal the rock mass strength. Only in extreme cases will the structural discontinuity determine the strength of a slope or a pillar. In most cases, it is an interaction of the two components in a rock mass which determines deformation and strength.

While assessing the effect of structural discontinuities in a slope wall or a pillar we have to find out to what extent movement is possible, for example, along an existing fault. If structural defects have carved out a block of rock which can move into an opening, these structural features will determine the stability of this block of rock. This case can readily be analysed by graphical methods (16).

In most cases, however, the structural defects do not separate a block of rock completely from its surrounding rock material. In such a case, failure will involve a complex process difficult to assess by calculation. Initially, some movement will occur along the geological fractures. Then the material bridges, as points of high strength, will attract high stresses and will fail with time.

This process of progressive failure takes time and can be monitored, owing to the emission of seismic impulses during the breakdown of material resistance. The peak strength reached will be lower than the laboratory strength of the rock substance and higher than the strength calculated from the shear resistance existing along fracture planes. These two boundaries can be defined by testing both sound rock material and fractured rock material. The use of these data is demonstrated in section B, under Excavation stresses and failure of rock.

B. Stresses in rock masses

Measurement of in situ stresses

Surface and underground excavations will fail if existing stresses exceed the strength of the rock forming the sides or the roof of the excavation. Stresses before any excavation are due to overburden weight and tectonic stresses. They

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can be quite high and an attempt should be made to estimate, even to measure them. All rock in the ground is compressed due to the surrounding stresses, so removal of the compression on a sample of rock will permit expansion due to the elastic behaviour of the rock. Stress relief of rock samples is easily accomplished by diamond drilling. Measuring the expansion of the rock during stress relief and knowing the elastic properties of the rock allows calculation of the stresses. The deformations observed are minute, and the instruments must be able to measure strains in the vicinity of 50 microstrain or deformations to the nearest 0.0005 mm. This can be achieved only by using electrical transducers; these instruments are delicate and must be handled carefully. Commercially available instruments are shown in figures 20, 21 and 22. To calculate the stresses from the measured strains, the elastic modulus must be determined with similar accuracy.

In underground situations, we are dealing with stresses in three dimensions, therefore the strain relief must be obtained in three independent directions. The resulting stress-tensor analysis is best done by computer because of its complexity (17, 18). All the calculations are based on elastic behaviour of rock, and measurements should be attempted only if the locality in which the measurements are made is free of fractures. A compilation of stress determinations from various areas of the earth crust (19) resulted in the following relationship of vertical stress with depth (figure 23):

$$\sigma_V = 19\ \mathrm{kg/cm^2} + 0{,}266\ \mathrm{kg/cm^2\ /m.}$$

This has been proven to a depth of about 2400 m and can be explained by the weight of the overburden.

In regard to horizontal components the situation is more complex and horizontal stresses can be found which are lower, equal to, and higher than the vertical stress. The surprising result is that actually 75 per cent of the compiled data in figure 24 indicate a higher horizontal than vertical stress, as given in the relationship of average horizontal stress and depth:

$$\sigma_H = 83\ \mathrm{kg/cm^2} + 0{,}\ 407\ \mathrm{kg/cm^2\ /m.}$$

This relationship is valid to a depth of 800 m. At greater depth, measurements are scarce, and predictions can be based either on the above formula or on assuming the horizontal stresses to be equal to the vertical stresses.

Horizontal stresses higher than vertical stresses cannot be explained by the weight of the overburden, so other stressing is indicated.

A possible explanation is offered by the unloading processing occurring during erosion, assuming rigid lateral confinement (20). Under these conditions, stresses in a horizontal direction will be reduced by less than stresses in a vertical direction:

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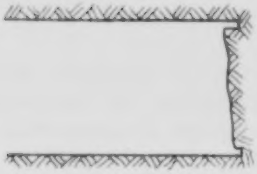
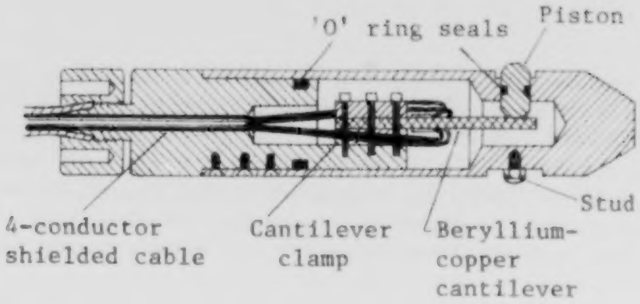
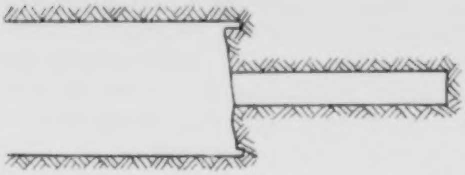


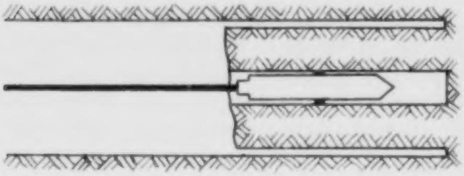
a) 15 cm hole drilled to required depth.


'0' ring seals
Piston
Stud
4-conductor shielded cable
Cantilever clamp
Beryllium-copper cantilever

U.S. Bureau of Mines Borehole Deformation Gauge.



b) EX hole drilled 30 cm beyond (diam 0.038 m)



c) Deformation meter overcored.

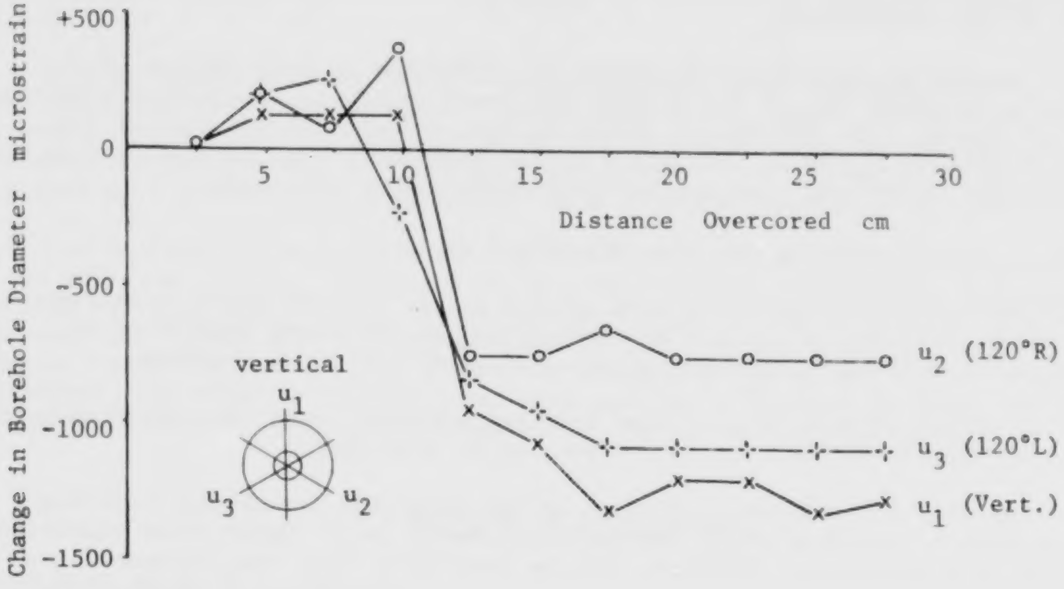
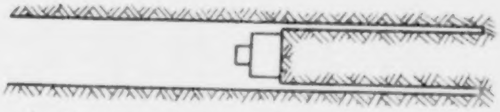


Figure 20. Overcoring Technique and Example of Measurements with the Deformation Meter

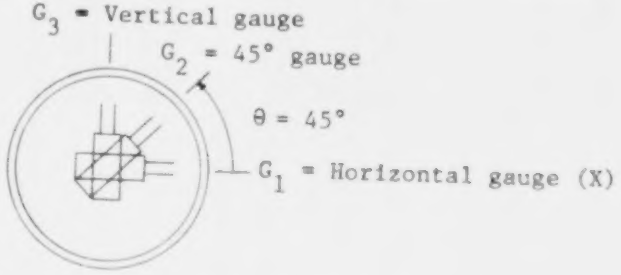
a) BX hole drilled to required depth,
flattened and polished (diam 0.06 m).

b) Strain cell bonded to end of hole.

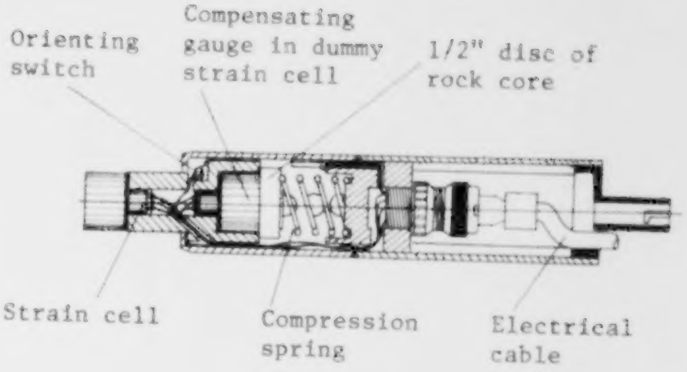


c) Overcored strain cell.

d) Recovered test piece with strain cell.


G_3 = Vertical gauge
G_2 = 45° gauge
θ = 45°
G_1 = Horizontal gauge (X)

e) Strain cell with stacked wire gauges.


Orienting switch
Compensating gauge in dummy strain cell
1/2" disc of rock core
Strain cell
Compression spring
Electrical cable

Installing Tool and Strain Cell

Figure 21. Geometry and Test Equipment of C.S.I.R. Strain Cell (Doorstopper) Overcoring Technique

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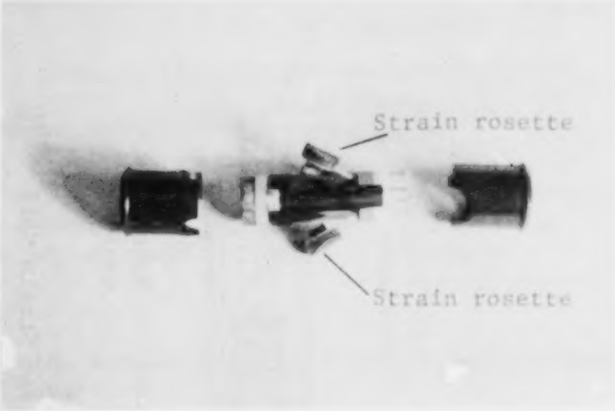
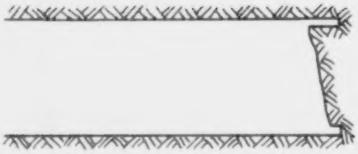
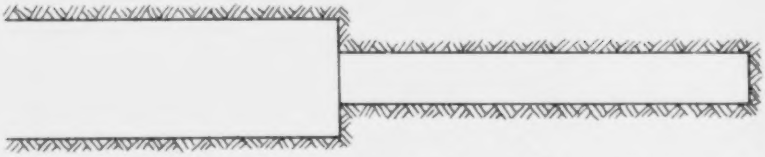

Strain rosette
Strain rosette

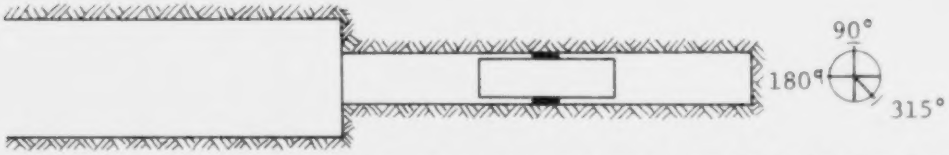
Triaxial Strain Cell which is Glued to the Inside of EX Hole.



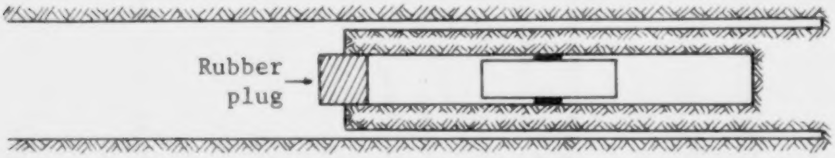
a) NXCU borehole drilled to required depth (diam 0.089 m).



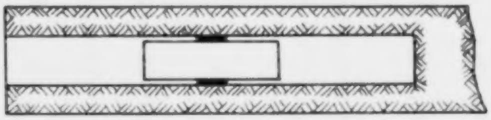
b) End of NXCU hole flattened and an EX borehole drilled for 45 cm.



c) Strain cell with three rosette gauges bonded to EX borehole (diam 0.038 m).


Rubber plug

d) Strain cell overcored.



e) Recovered test piece with strain cell.

(not to scale)

Figure 22. Geometry for C.S.I.R. Triaxial Strain Cell Overcoring Technique

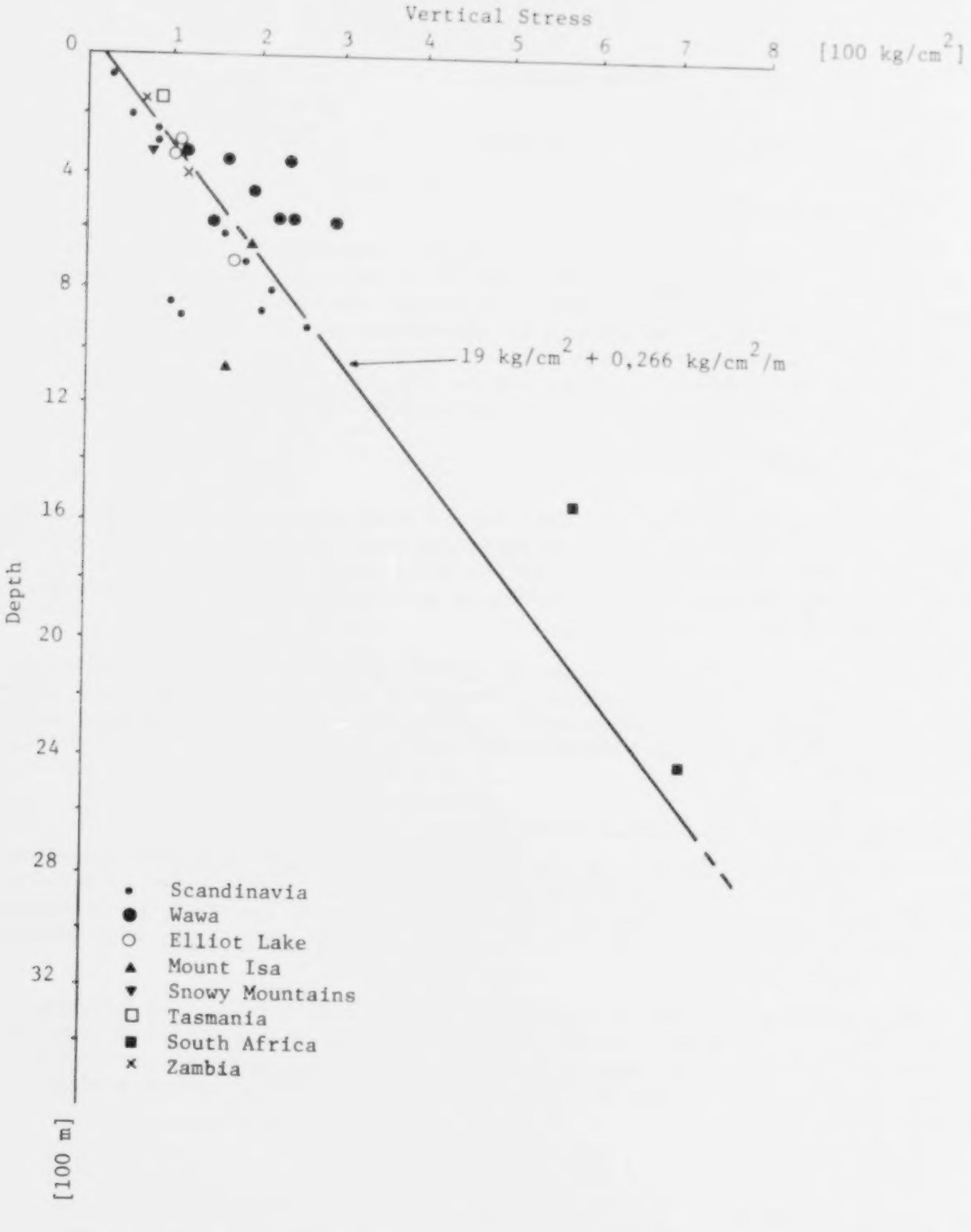


Figure 23. Increase of Vertical Stress with Depth

$\Delta\sigma_H = \left[\frac{\upsilon}{1 - \upsilon} \right] \Delta\sigma_V$, where

$\Delta\sigma_V$ = reduction of vertical stress,

$\Delta\sigma_H$ = reduction of horizontal stress,

υ = Poisson's ratio.

Thus during erosion, the horizontal stress component will increase relatively to the vertical stress component and, even if at depth the vertical component is larger than the horizontal one, this relationship can be reversed during unloading. In figure 24, an example is plotted for the average horizontal stress component of a block of rock subjected to a hydrostatic stress field at a depth of 2,400 m and then unloaded by removing all the overburden. Assuming a Poisson's ratio of 0.25, the stress at the surface would be:

$\sigma_H = 445$ kg/cm^2 and $\sigma_V = 0$.

Using relationship 3 as the limit stress condition, one finds that horizontal stresses of this magnitude cannot be sustained over a long term, and that fracturing occurs at a depth of about 1,100 m for the given example. Below this depth, a failure condition is not reached and, therefore, at greater depth a smaller horizontal stress gradient may exist.

Stress determinations are based on elastic theory and can be successfully made only in rock with elastic properties. Fractured and broken rock are poor subjects for doing stress measurements. In general, stress determinations are complex and beyond the capabilities of most mining companies and should be contracted out to research organizations.

Excavation stresses and failure of rock

With data available on rock strength and some results available on ground stresses, the next step is to calculate stress changes caused by an excavation and to obtain the loads acting on rock forming essential parts of an underground or surface structure. However, not all mining methods require the same emphasis in this respect.

Underground mining can be subdivided into methods which aim at inducing failure (longwall, sublevel caving, block caving) and methods which attempt to prevent failure and to use rock for support purposes (room-and-pillar). The tonnage percentages indicate the importance of the various Canadian mining methods (21).

Surface mining	68%
Support methods	25%
Shrinkage stoping, cut-and-fill and room-and-pillar caving methods	7%

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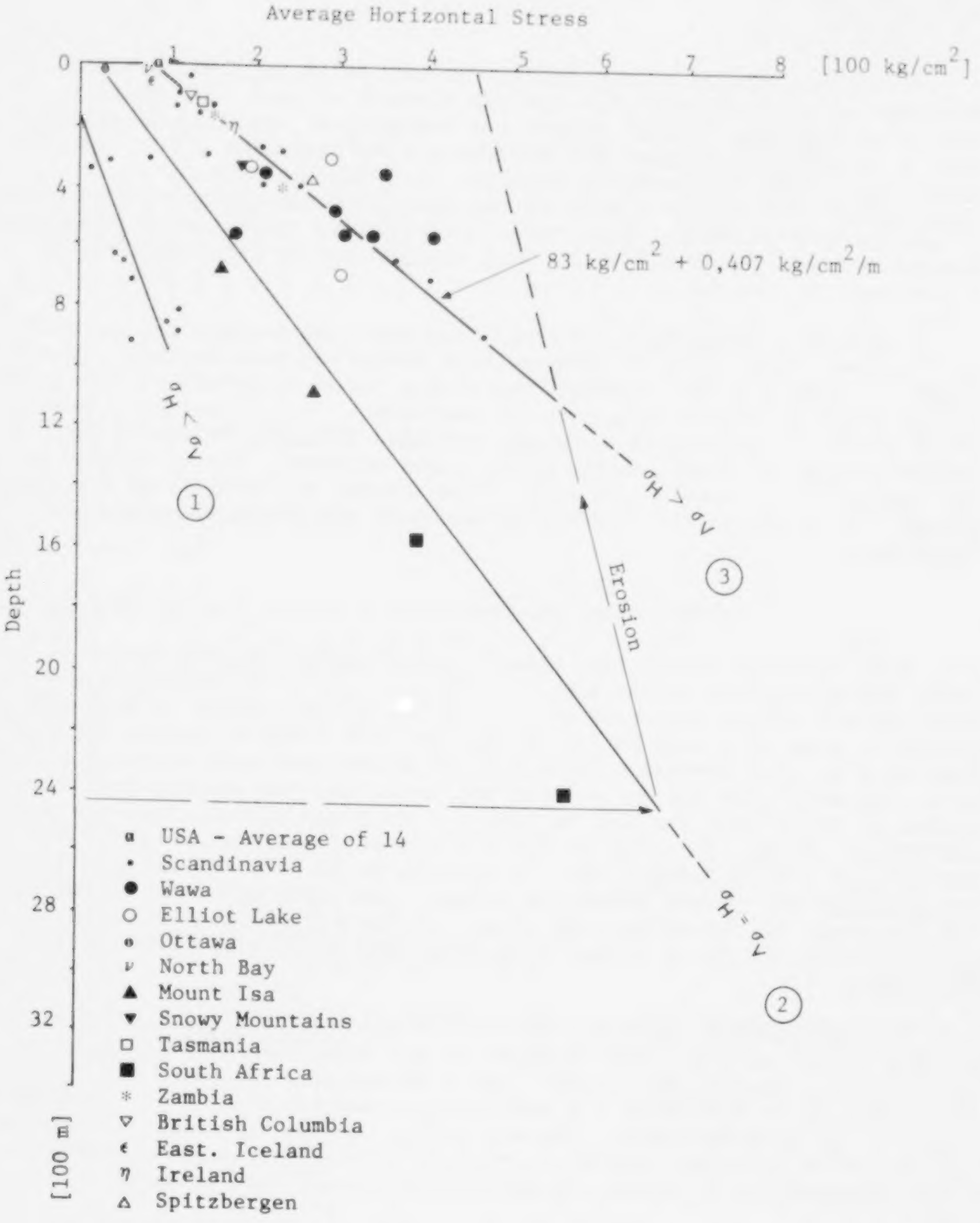

Figure 24. Increase of Average Horizontal Stress with Depth

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Preventing failure of rock pillars or rock walls requires detailed knowledge of excavation stresses and the strength of rock. In regard to excavation stresses, various methods are available and their applicability depends on the complexity of the geometry and the variation of the rock properties. Exact mathematical solutions are available only for two-dimensional cases of simple geometry and rock properties which stay strictly within the elastic range. Thus stress distributions around single underground openings can be readily obtained. Many examples can be obtained from the literature (22, 23, 24).

Interaction of openings and irregular stopes can be modelled and analysed by the photo-elastic method; however, this method has been replaced, to a large extent, by finite-element analysis which is a numerical method of calculating stress distributions for a variety of conditions. In this method, a section which is to be investigated is broken down into triangular elements which are joined at their vertices, called nodal points or nodes. Forces applied on the outside of the section are transmitted through the elements at their nodal points. The elements are fitted with rock-like properties by using the relationship,

$$\text{Stiffness} = \frac{\text{Force}}{\text{Deformation}} .$$

This stiffness coefficient allows the calculation of deflections at the nodal points from the forces applied to the outside of an excavation section. From the deflections, stresses and strains can be calculated. A typical result is given in figure 25 for an open pit (25). Finite element methods have been used to great advantage in laying out underground power stations. In a mine, the excavation has to be where the ore is and this results in a lesser degree of flexibility for the designer but can yield valuable data for stope and pillar layout. The finite-element analysis is the most promising tool at the present time to analyse stress distributions in rock and can go a long way in modelling the various excavation stages. The cases discussed so far are all solutions for two-dimensional cases. A three-dimensional analysis is possible with the finite element method and with photo-elasticity but both are difficult.

The mathematical approach, the photo-elastic method, and the finite-element method provide exact solutions in the sense that at every point in the studied cross-section the stresses can be determined. However, this type of perfection is in many cases not justified because the rock is not sufficiently known or the load/deformation characteristics are too variable. Then it is sufficient to calculate average stresses or strains for essential parts of the excavation and to select the most suitable excavation geometry.

In room-and-pillar mining, the best-known method of estimating pillar stresses is the tributary area method. Pillar stresses are calculated on the basis that the pillar has to support all the ground overlying an adjacent (tributary) excavation. It can be used only in good-quality rock. If a sequence of rooms and pillars exists the average stress in the pillar is:

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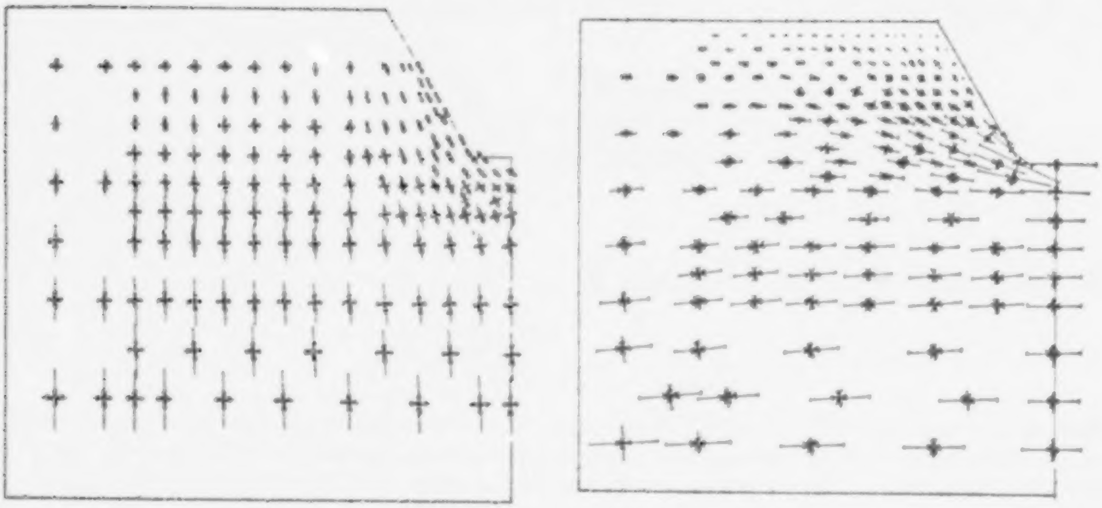
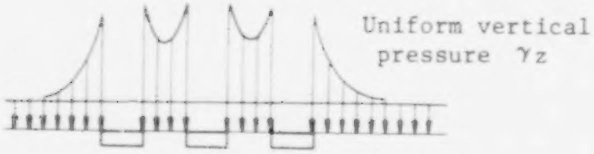


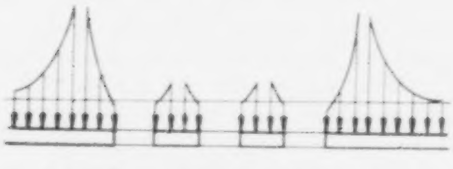
(a)

(b)

Figure 25. Stress Trajectories from Finite-Element Models for (a) k = 0.33 and (b) k = 3. (k = horizontal stress/vertical stress) After Coates, 1970



(a) Distribution of pressure on pillars in a homogeneous, elastic material.



(b) Distribution of pressure when pillars yield by fracturing.

Figure 26. Distribution of Pressure on (a) Elastic and (b) Yielding Pillars

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$$\sigma_p = S_z \, A_t / A_p \ ,$$

where,

S_z = vertical field stress,

A_t = area of an excavation tributary to a pillar,

A_p = area of pillar.

An expression of the pillar stress σ_p in regard to extraction ratio R is:

$$\sigma_p = \frac{S_z}{(1 - R)}$$

For an extraction ratio of 100 per cent the stresses approach infinity. Calculations with this formula can be regarded only as a first approximation and lead in many cases to conservative designs. An example is given for a mine pillar in siderite (figure 27). The strength was measured by testing continuous samples of siderite (height to width ratio 2:1) with diameters between 2 to 25 cm. This showed clearly a decrease in strength with size. Samples containing open geological fractures were similarly tested for compressive strength. There the uniaxial compressive strength increased with size. Because further changes of the strength of rock with increase in size could be anticipated, the strength of pillars in sound siderite was estimated to be between the strengths of the broken and sound samples. This value and the standard deviation of the sound samples was taken as representative of a pillar in sound siderite (810 ± 316) kg/cm^2. On the same level, stress measurements, carried out at four locations, indicated a mean value of 306 kg/cm^2 for the stresses in horizontal direction. According to the tributary area method, for a 67 per cent recovery ratio, the mean stresses on the pillar would be three times the in situ stress. Plotting the cumulative frequency curves for these values showed (figure 27) that the probability of failure is 65 per cent, which means the pillars will be fractured.

Mining experience has shown, however, that even a pillar which is loaded beyond its capacity can still provide local support (26). In this case, rocks in the vicinity have to take up the excess loads. This principle is illustrated in figure 26, where the pressure distribution is given for homogeneous elastic rock and for pillars which yielded by fracturing. Pillars which have exceeded their peak strength can still support large loads as indicated by post-failure strength of rock samples (figure 16). But, as the post-failure strength of rock sample depends on the type of testing machine, so does the post-failure strength of a pillar depend on the ability of the surrounding rock mass to react as a structural unit. Successful application of the method requires that "arching" or interlock can occur above a mine area so that a large proportion of the weight of the undermined rock can be transferred to abutments. Most of the experience is available from coal and potash mines at depths of about 600 metres, which experienced poor roof and floor conditions during mining. A reduction of pillar sizes allowed the pillars to yield slightly and roof and floor conditions improved. This type of method is the only one which offers application of the room-and-pillar method at great depth. No simple numerical approach is available, and choosing this method requires a study of comparable mining situations and a great deal of experimenting and monitoring in the initial mining

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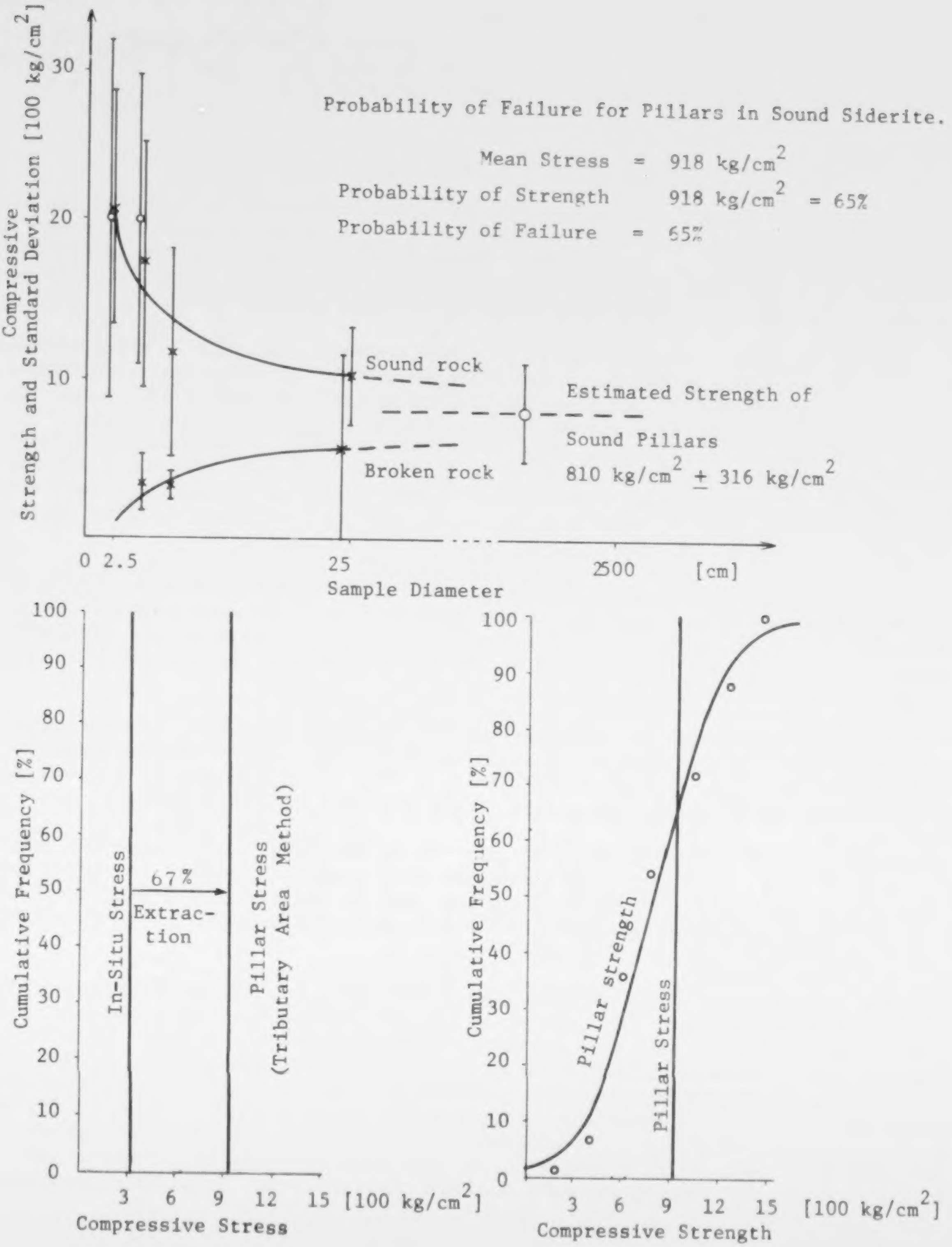

Figure 27. Probability of Failure for Pillar in Sound Siderite Calculated from In-Situ Stresses and Laboratory Strength Tests on Samples of Different Size

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phase. Pillars should fracture but not fall apart, so a delicate balance must be maintained. The knowledge of ground stresses, in situ rock strength, in conjunction with rock deformation, are essential to achieving this objective.

C. Deformation monitoring in rock

Analyses of strength and stress are the classical methods of defining failure of a rock mass. However, the visual evidence of failure is the deformation or movement which has taken place. On an empirical basis it is sometimes possible to evaluate the stability of a rock mass by measuring deformation. Fortunately deformation is one of the easier parameters to measure by means of borehole extensometers.

Figure 30 shows a typical layout of a multi-wire installation. Wires attached to anchors are installed at various depths in a borehole. An extensometer connected to a station at the borehole collar measures the position of the end of each wire. In other words, the wire is used as a standard gauge length and the extensometer measures the change in length of the borehole relative to this standard. The axial strain between adjacent anchors is calculated as shown in figure 30. In this way, the magnitude, rate, and location of movement can be determined. Strain rate is probably the most sensitive indicator of stability, an accelerating strain rate indicating approaching instability and either a constant or a zero strain rate indicating stability.

Figure 31 shows an example of extensometer measurements taken in a pillar (27). Most of the expansion was confined to the outside 4 m of the west side and 3 m of the east side of the pillar as the adjacent stopes and pillars were mined. This showed that fracturing was confined to the edges of the pillar, whereas the central 15 m remained relatively undisturbed.

There are many types of extensometers, from simple mechanical devices which are read manually to sophisticated electrical transducers which can be read remotely. One disadvantage of borehole extensometers is that only axial movement of the borehole is measured, any movement at right angles to the borehole is not picked up.

In some cases it is necessary to measure the absolute three-dimensional movement of the rock mass. Such a case is the movement of open pit slopes. Surveying instruments are being increasingly used for this purpose and, in the last few years, there has been a rapid advance in electro-optical instruments which can measure distance to a fraction of a millimetre.

The general layout of this measuring system is to locate the surveying instrument on the pit crest opposite the slope which is considered potentially unstable. The instrument station is tied to a system of stable reference stations to ensure that it does not move. Several reflector targets are permanently attached to the opposite slope wall and the instrument measures the distance to these targets. With electro-optical instruments, it is also essential to measure the vertical angle to each of the targets so that the three-dimensional co-ordinates can be calculated. The major advantage of this system is that, after installation of the reflector targets, access is not required on the pit wall and the measurements are made from a remote, accessible location.

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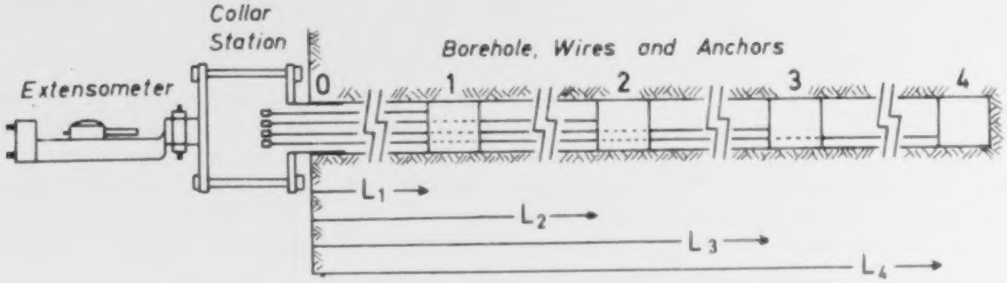
Collar Station
Extensometer
Borehole, Wires and Anchors
0
1
2
3
4
L₁
L₂
L₃
L₄

	Movement	Strain
Wire L_1	ΔL_1	$\varepsilon_{0,1} = \frac{\Delta L_1}{L_1}$
Wire L_2	ΔL_2	$\varepsilon_{1,2} = \frac{\Delta L_2 - \Delta L_1}{L_2 - L_1}$
Wire L_3	ΔL_3	$\varepsilon_{2,3} = \frac{\Delta L_3 - \Delta L_2}{L_3 - L_2}$
Wire L_4	ΔL_4	$\varepsilon_{3,4} = \frac{\Delta L_4 - \Delta L_3}{L_4 - L_3}$

Figure 30. Layout of Anchors and Wires in a Borehole

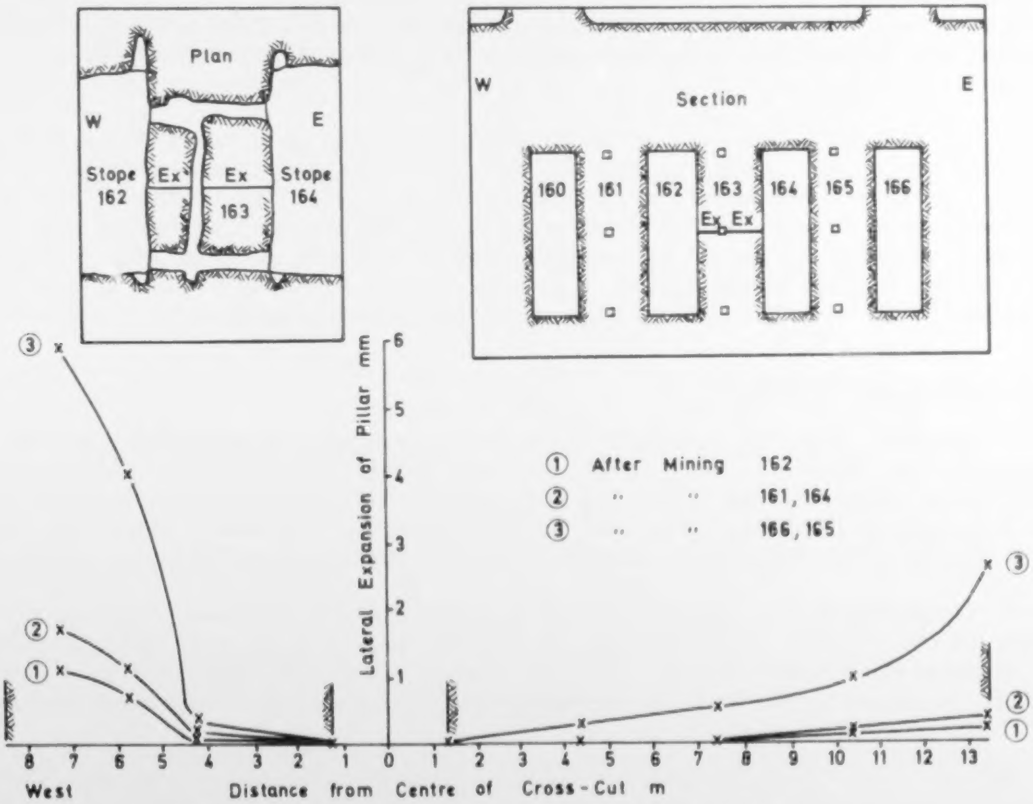

Figure 31. Example of Extensometer Measurements in a Pillar

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At the Mining Research Centre, three measuring systems are being tried out. At one open pit, a triangulation network has been established and a precise theodolite is being used to measure the angular movement on about 30 targets. At another open pit, a laser-theodolite will be tested. This instrument measures electrically the distance and the horizontal and vertical angles to the targets. Finally, at the same open pit, a photo-theodolite will be used to obtain stereo-photographs of the pit slope. From a series of photographs, the rock mass movement can be measured.

Figure 32 shows the horizontal movement measured with surveying instruments over 19 years across the San Andreas fault in California (28). This example shows clearly what can be achieved by surveying to establish patterns of movement for evaluating slope stability.

D. Practical applications

The following is a brief review of some of the applied rock mechanics research done in Canada and other parts of the world.

Rockbursts and pillar destressing

Rockbursts in some deep mines throughout the world are a major ground control problem. Research has been done for a number of years in the deep gold mines in South Africa (29, 30) and more recently in the Coeur d'Alene mining district of northern Idaho, United States (31, 32). In the Coeur d'Alene district, thin, irregular, and nearly vertical silver-lead veins are mined by a horizontal cut-and-fill method. Bursting usually occurs if the crown pillar thickness is less than 15 m. The United States Bureau of Mines installed a micro-seismic monitoring system in rockburst-prone stopes and showed that potential burst zones could be delineated weeks in advance of a burst.

One control measure is to destress the pillars by blast-induced fractures. This reduces the stress acting on the pillar, decreases its effective elastic modulus, and, hence, decreases the amount of energy which can be stored in the pillar. Figure 33 illustrates a seismic velocity survey done in a crown pillar before and after destressing Seismic velocity is related to elastic modulus of the rock and any decrease would indicat a yielding or less rigid pillar. As can be seen after destressing, the seismic velocity substantially decreased and subsequently the pillar was mined without undue difficulty.

Open-pit slope stability

Research into the stability of open-pit slopes has received a lot of attention in the last few years, because large economic benefits result from being able to increase the slope angle by only a few degrees. Multi-discipline investigations have been done by Imperial College in London and another study is presently under way in Canada under the direction of the Mining Research Centre, Department of Energy, Mines and Resources.

A comprehensive case history of a slope failure has been done at the Chuquicamata Mine in Chile (33). The open pit is 3.3 km long, 1.0 km wide and up to 400 m deep with a slope angle of 42° to 46°. Tension cracks were first observed on a section of the slop crest in August 1966, but the movement was very small and soon ceased. An earthquake in

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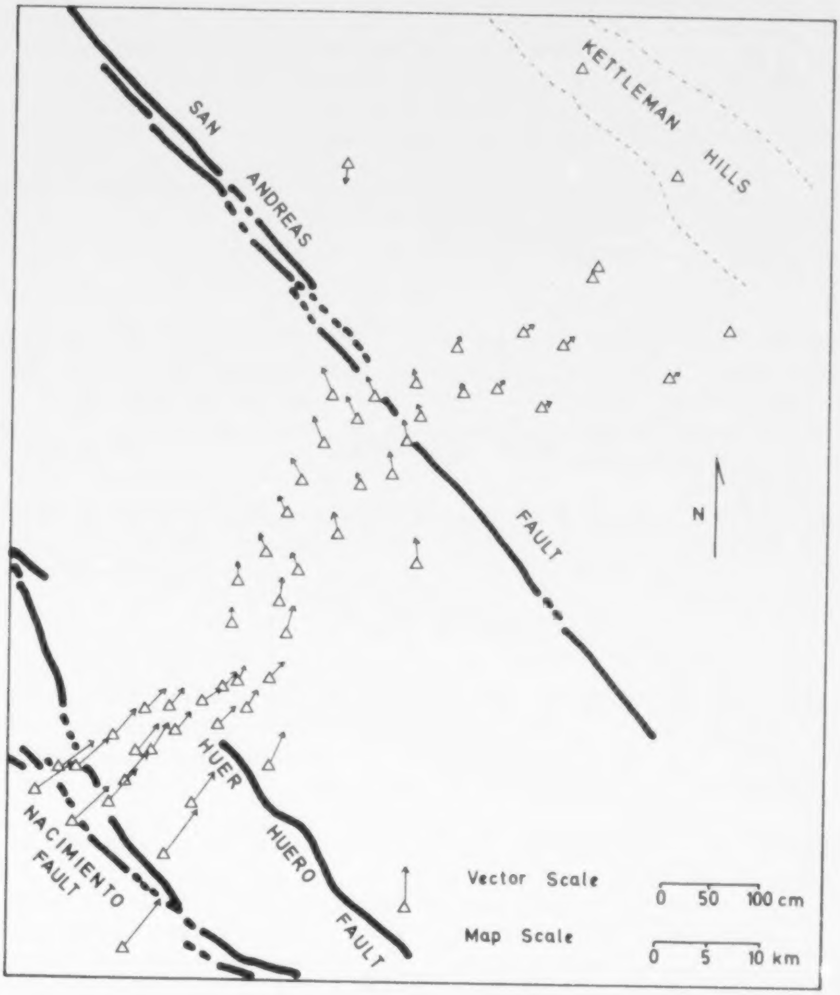
SAN ANDREAS FAULT
KETTLEMAN HILLS
N
HUER HUERO FAULT
NACIMIENTO FAULT
Vector Scale
0 50 100 cm
Map Scale
0 5 10 km

Figure 32. Horizontal Movement between 1932 and 1951 in Vicinity of the San Andreas Fault (Burford, 1965)

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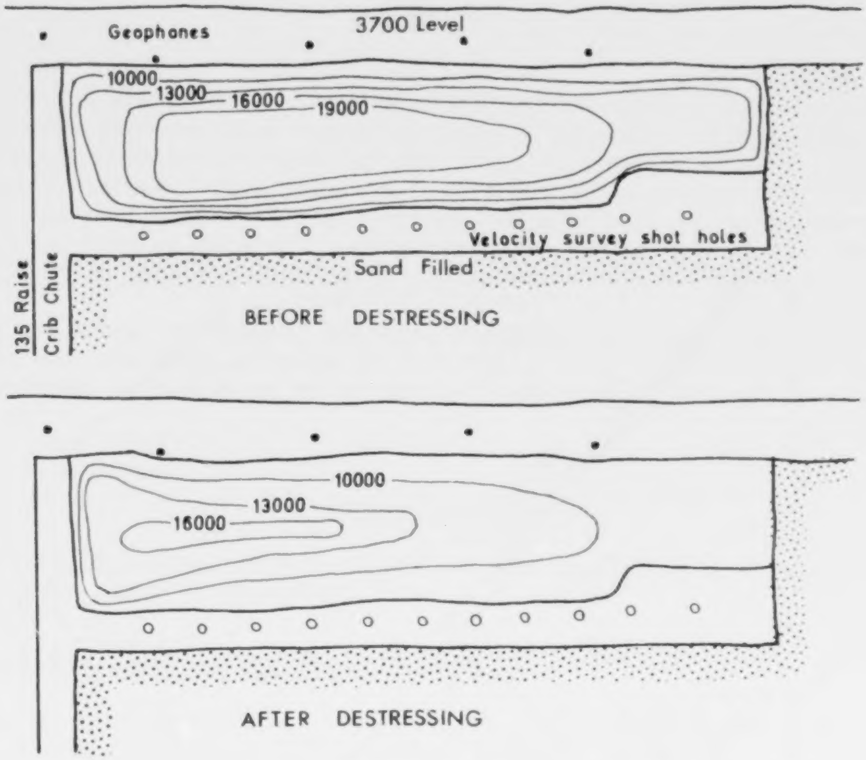
Geophones
3700 Level
10000
13000
16000
19000
Velocity survey shot holes
Sand Filled
135 Raise
Crib Chute
BEFORE DESTRESSING
10000
13000
16000
AFTER DESTRESSING

Figure 33. Contours of Seismic Velocity before and after Destressing (Blake, 1971

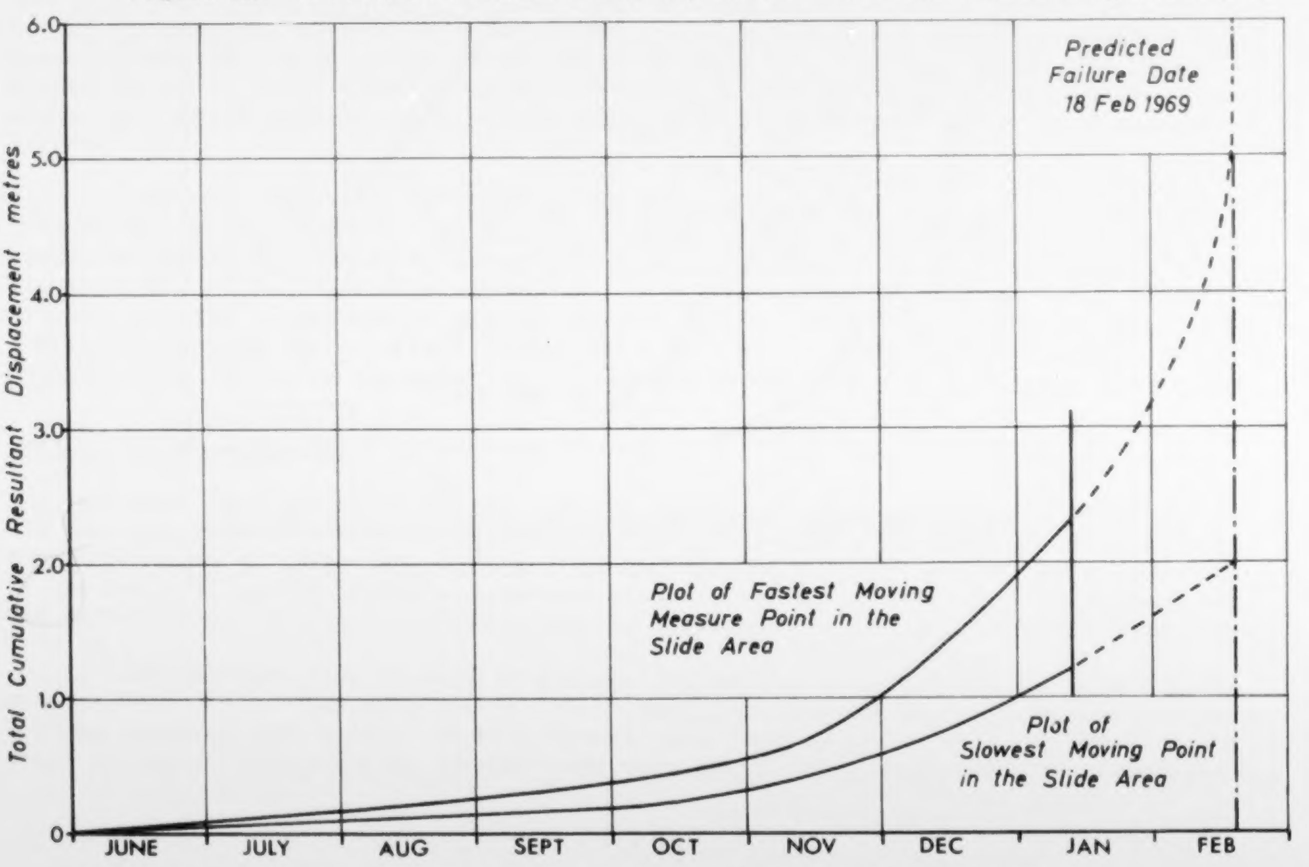

Figure 34. Movement of a Pit Slope Leading to Failure (Kennedy, 1971)

December 1967 reactivated the movement along these cracks. Numerous instruments, including crack extensometers, tape and transit line systems, quadrilateral-tension crack extensometers and seismographs, were then installed on the slope. In January 1968, movements between 2 and 5 mm/day were measured on the tension cracks and between 1 and 3 mm/day on the slope face.

Geological surveys and exploratory drilling were used to define the area of the potential slide. An unloading programme was started at the top of the slide area in April 1968 and 4.5 million tons was removed. In November 1968 a blast at the toe of the slope caused a considerable acceleration in movement to between 2 and 7 cm/day. To minimize stoppage of operations it was necessary to predict a date for slope failure. The fastest-moving point on the slope face was chosen for extrapolation and, from experience and engineering judgement, it was estimated that this point could tolerate about 6 m displacement prior to failure. Figure 34 shows the displacement recorded up to 13 January 1969 and the predicted displacement up to failure on approximately 18 February 1969.

The displacements increased to between 19 and 46 cm/day on 11 February and small rock falls occurred. The mine closed down on 16 February owing to the hazardous conditions. On 18 February the crest was moving at 136 cm/hour and the slide occurred on the same day (as predicted). The slide involved 12 to 15 million tons of rock. Full production was resumed on 19 February.

This type of study illustrates what can be achieved to minimize the effects of such a large slope failure. However, it should be emphasized that not every slope failure can be predicted so accurately.

Pillar design

In many instances, empirical design criteria can be obtained by examining and analysing the stable and unstable conditions already existing in mines. In this way, instead of using trial-and-error methods in a new mining area, the stability of rooms, pillars and other mine structures is extrapolated from previous experience.

This type of approach was used to estimate the strength of coal pillars in South Africa (34, 35). A comprehensive survey was done in the collieries and information was obtained on 98 stable and 27 collapsed pillar geometries. The data collected included depth below surface, height and width of square pillars, and room width.

The concept of a safety factor was used to analyse the data. Safety factor is defined as the ratio of pillar strength to stress acting on the pillar; a value of more than 1 indicates stability and less than 1, instability. It was realized that the information on pillar dimensions was not exact and that pillar stress could only be estimated. Consequently, a statistical approach was used based on the assumption that the collapsed pillars would be normally distributed around a safety factor of 1.

The stress acting on the pillars was estimated from the weight of the overburden which is taken as (0.25 kg/sq cm) x metres of depth and the extraction ratio as follows:

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$$\text{Pillar stress} = \frac{0.25\ D}{1 - R},$$

where pillar stress is in kg/sq cm, D = depth metres, R = extraction ratio. Pillar strength, as related to pillar height (h) and width (w), was calculated so that the collapsed pillars were clustered around a safety factor of 1, and the stable pillars had a safety factor greater than 1.

$$\text{Pillar strength} = 72.5\ \frac{w^{0.46}}{h^{0.66}},$$

where the strength is in kg/sq cm, 72.5 kg/sq cm is the strength of a metre cube of coal, and pillar width and height are in metres.

Figure 35 shows the estimated pillar stresses and strengths for the collapsed and stable pillars. There is good agreement between the analysis and the observed pillar conditions in that the collapsed pillars are grouped around a safety factor of 1 and all except one of the stable pillars have safety factors greater than 1. This gives confidence in applying this method of analysis to design optimum pillar dimensions.

A similar study has been made in the uranium mines at Elliot Lake, Ontario, Canada (36). Figure 36 shows the location of 13 mines, all except one being brought into production between 1954 and 1957. At present, two mines are producing. The orebody is in the form of a syncline and is relatively flat. A system of stopes and pillars was used at all the mines, and a typical layout is shown in figure 37. As mining progressed, stope-and-pillar dimensions were modified to obtain greater extraction, and, if pillars or roofs failed, the extraction was reduced to provide more stable conditions.

Information was gathered from the producing and closed-down mines on mining dimensions and cases of stable and collapsed pillar geometries. The depth below surface for the 13 mines ranges from zero to 1050 metres. The dip of the orebody varies considerably due to local rolls but averages about 20° between 0 and 900 metres and about 10° below this. Dimensions of the stopes and pillars have varied considerably in the past but present practice is fairly uniform, with 20-metre stope spans extending about 75 metres on dip. Mining height is between 2 and 6 metres with pillar height and width usually maintained at a 1:1 ratio. Extraction has ranged from 60 to 90 per cent with, in general, the greatest extraction being obtained in the shallowest mines and the least extraction in the deepest mines.

Information was obtained on three cases of collapsed pillars, three cases of partially failed pillars (appreciably spalled on the sides), and 23 cases of stable pillars. Owing to the limited number of collapsed pillars, a statistical analysis, similar to that done for the coal pillars in South Africa, could not be done. Instead, the strengths of the pillars were estimated from compressive strength tests and the stresses acting on the collapsed pillars. The stresses acting on the pillars were estimated from in situ stress measurements (similar to those described earlier in the paper) and the weight of the overburden.

Estimated pillar stress is plotted against pillar strength for all the pillar cases in figure 38; in addition, lines representing safety factors from 1.0 to 3.5 are drawn. It can be seen that the results for the crushed, partially failed and stable pillars are comparable with the observations. The safety factors of crushed pillars are grouped

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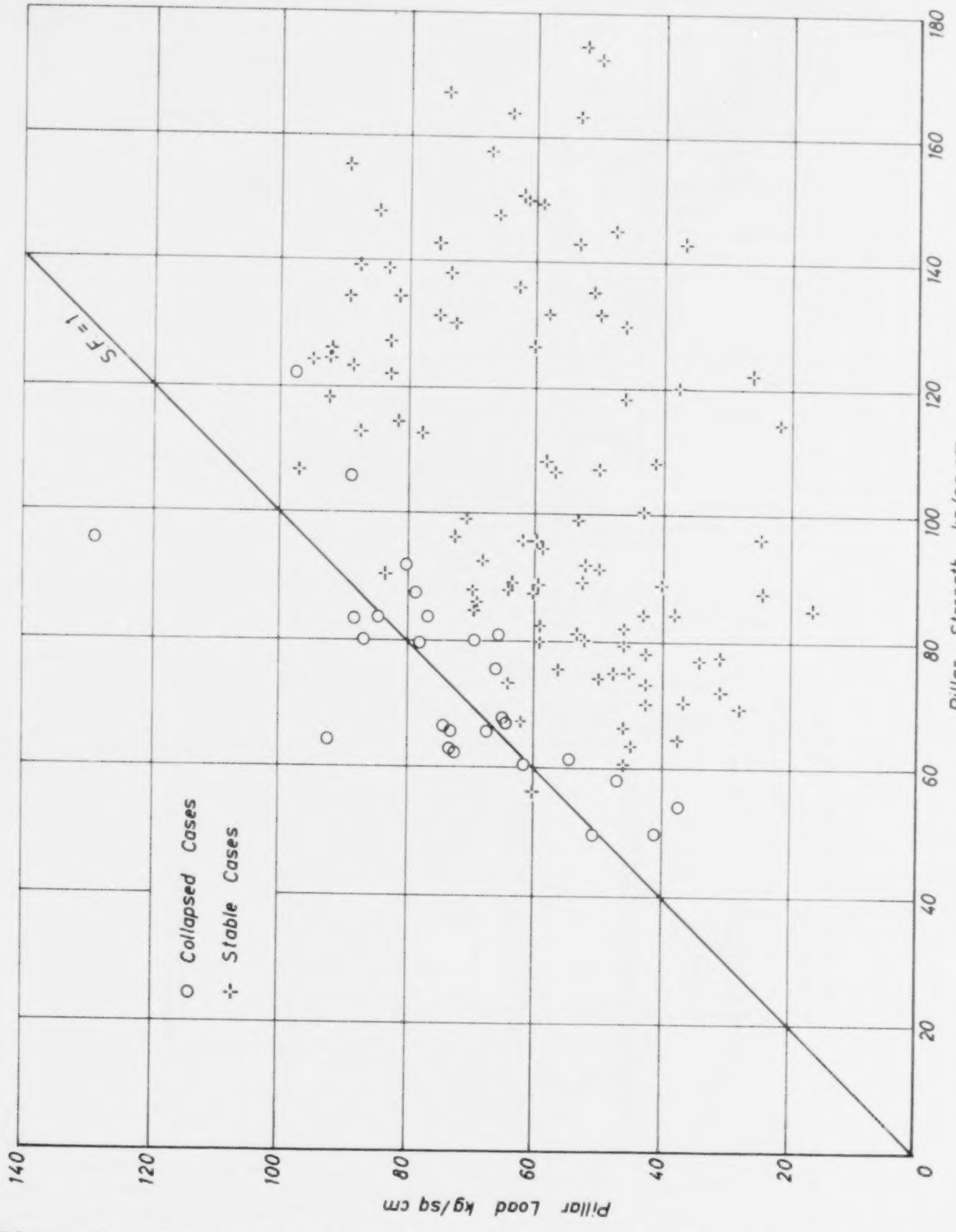

Figure 35. Observed Pillar Loads as a Function of Calculated Pillar Strengths for Collapsed and Stable Areas of Mining (Salamon, 1966)

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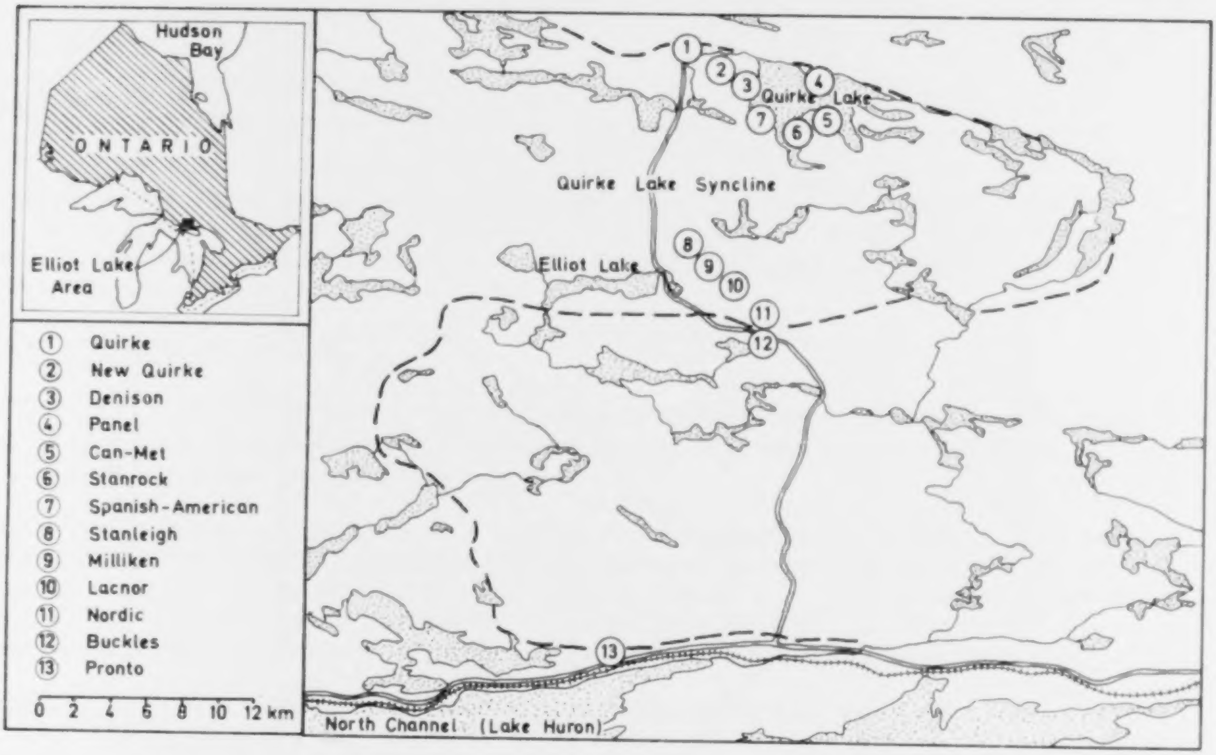
Hudson Bay
ONTARIO
Elliot Lake Area
Quirke Lake
Quirke Lake Syncline
Elliot Lake
① Quirke
② New Quirke
③ Denison
④ Panel
⑤ Can-Met
⑥ Stanrock
⑦ Spanish-American
⑧ Stanleigh
⑨ Milliken
⑩ Lacnor
⑪ Nordic
⑫ Buckles
⑬ Pronto
0 2 4 6 8 10 12 km
North Channel (Lake Huron)

Figure 36. Location of Uranium Mines in the Elliot Lake Area

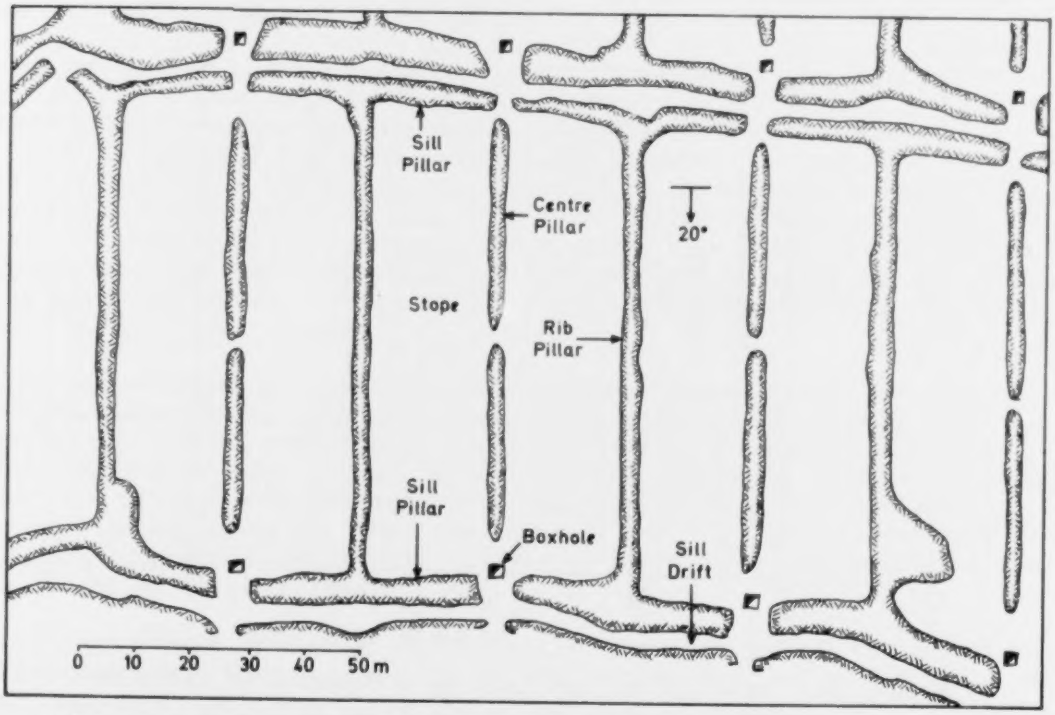
Sill Pillar
Centre Pillar
20°
Stope
Rib Pillar
Sill Pillar
Boxhole
Sill Drift
0 10 20 30 40 50 m

Figure 37. Typical Layout of Stopes and Pillars in the Elliot Lake Mines

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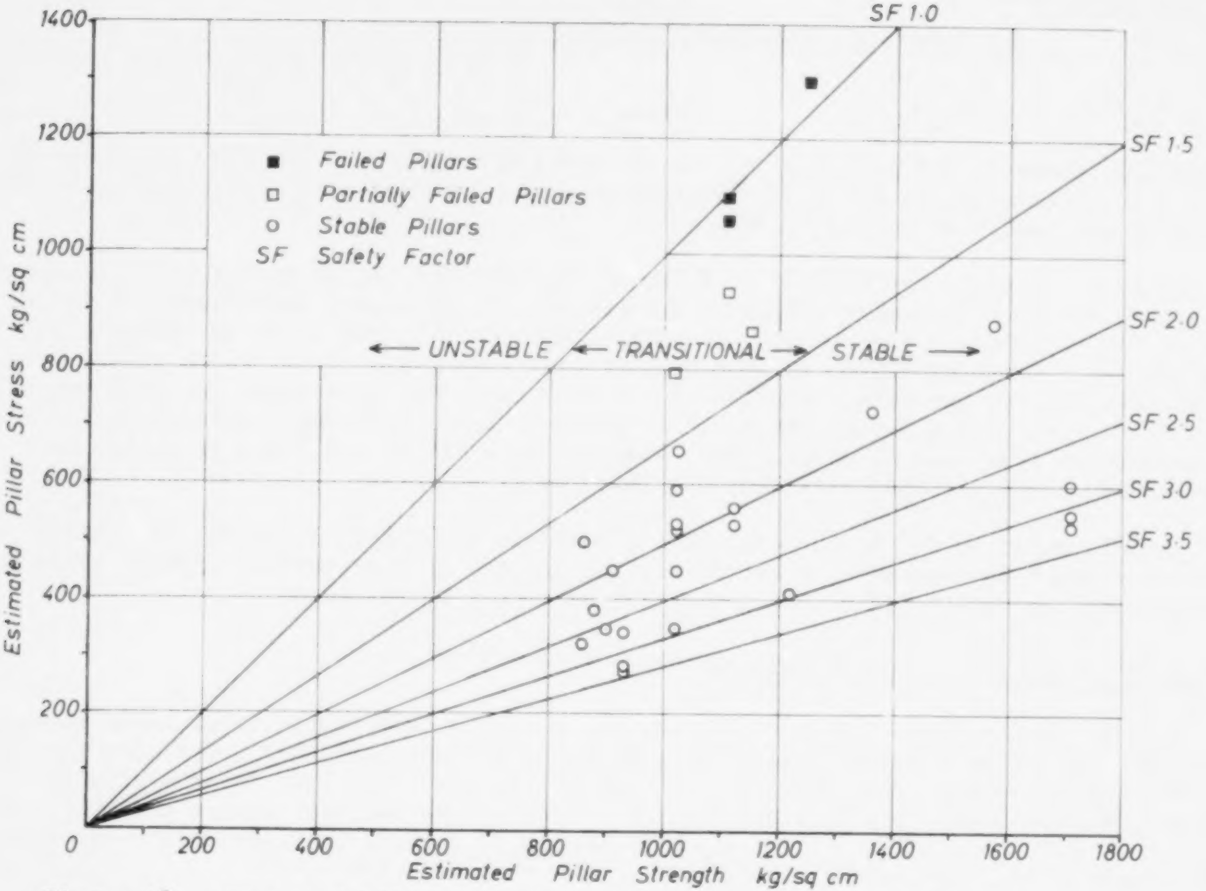

Figure 38. Estimated Pillar Stresses and Strengths, Elliot Lake

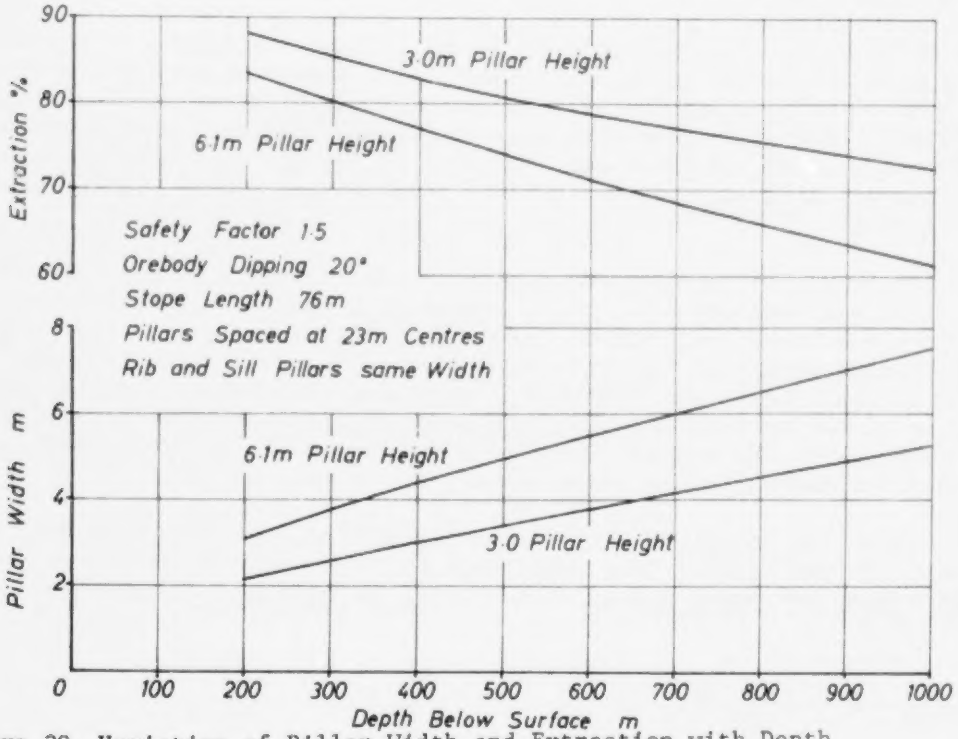

Figure 39. Variation of Pillar Width and Extraction with Depth

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around 1.0, those of the partially failed pillars lie between 1.0 and 1.5, and those of the stable pillars all exceed 1.5. Hence there is no contradiction between the calculated stability and that observed underground.

These results can now be used to determine the minimum pillar widths and correspondin extraction as depth increases. Figure 39 shows the pillar widths and extraction for an average stope-and-pillar layout with a safety factor of 1.5, and 3.0-m and 6.1-m pillar heights. At a depth of 500 m, a 3.0-m high pillar, requires a pillar width of 3.4 m for an extraction of 81 per cent. A pillar 6.1 m high requires 5.0 m width for a 74 per cent extraction. At 1,000 m depth, the width of a 3.0-m high pillar has increased to 5.3 m and the extraction decreased to 72 per cent, whereas the width of a 6.1-m high pillar has increased to 7.5 m and the extraction decreased to 61 per cent.

This is about as far as the engineering analysis on pillar design can be taken. Extraction might be increased by extracting the pillars on a retreating system. However, in this case the experience of the mining engineer must be relied on.

Design of an artificial support system

Rock bolts have been used for a number of years to support the local loose around mine openings. More recently, longer and stronger cable or solid bar bolts have been used to support larger areas of ground in open pit slopes (37), the roof above large stopes (38), and around ore pass systems (39). The Mining Research Centre recently co-operated with a Canadian salt mine in the design of a cable bolt system for the salt roof strata above a new underground crushing and screening plant (40).

Roof falls have been a problem at this room-and-pillar mine since the start of operations. Initially these falls were thin slabs, up to 0.5 m thick, but the use of 1.2-m long roof bolts prevented falls of this type. Later, despite the use of these bolts, heavier falls occurred up to 2.4 m thick and in some instances, if allowed to continue, penetrated the overlying dolomite about 7.7 m above the mine roof. A reduction in room width from 18 to 14 m reduced but did not entirely eliminate the occurrence of the heavier roof falls. This experience with roof conditions suggested that additional support was necessary to ensure long-term stability in the new mill area. A civil engineering method was used, with post-tensioned steel cables grouted in boreholes drilled into competent ground.

To design the layout of this support system it was necessary to know the volume of strata requiring support and the location of competent ground to anchor the cables. This in turn determines the number and support capacity of the cable bolts. Roof deformation measurements in conjunction with the experience of existing roof falls indicated that a 2.4-m thickness of roof strata was potentially unstable. Expansion measured above the central half-span of the roof showed that this area was unsuitable for anchoring the cable bolts. Compression was measured in the roof strata over the pillar edge; and this area was suitable for anchoring the cable bolts.

Figure 40 shows the layout of the cable bolts and the potential roof fall which requires support. It was decided that four cable bolts per row, spaced at 2.7 m centres, and drilled upwards at 45° and 9 m long would fulfil the design requirements. The

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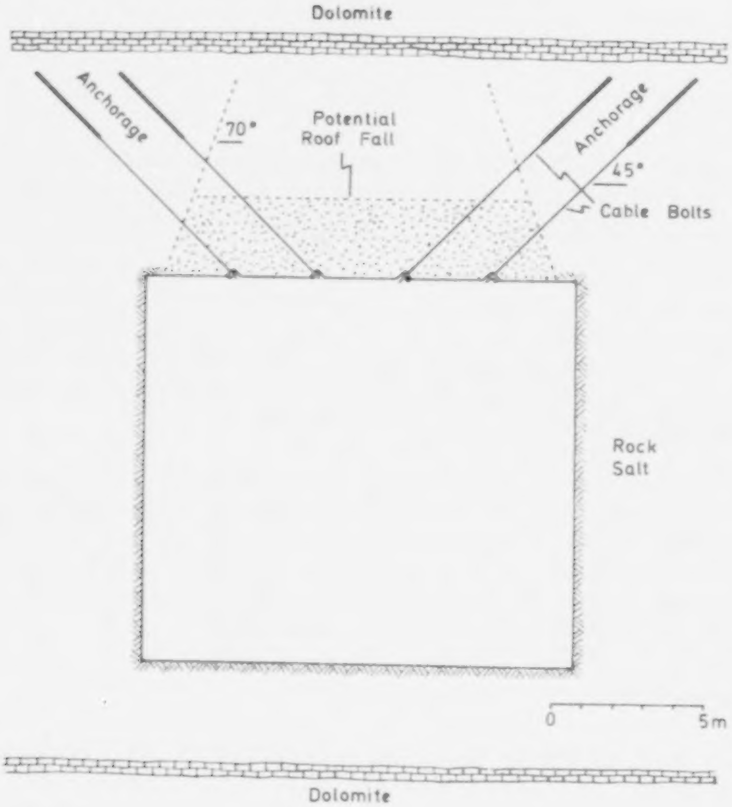
Dolomite
Anchorage
70°
Potential
Roof Fall
Anchorage
45°
Cable Bolts
Rock
Salt
0
5m
Dolomite

Figure 40. Design and Layout of Cable Bolts.

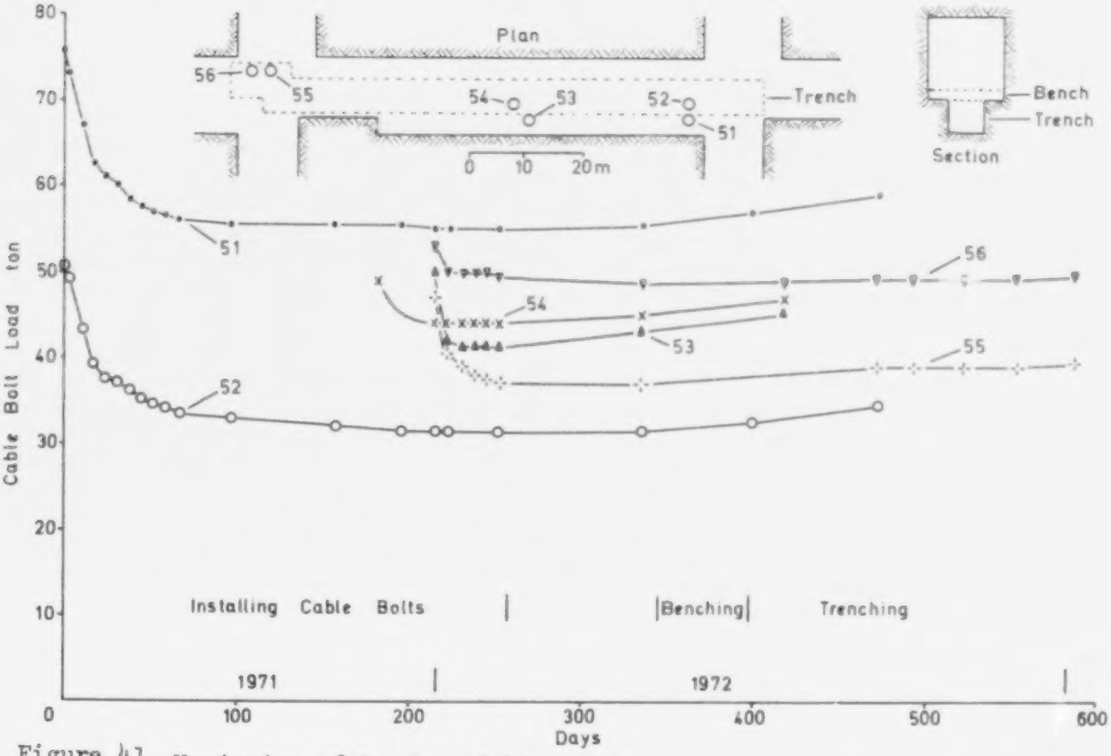

Figure 41. Variation of Load on Cable Bolts

distance between rows of bolts is 2.7 m and it can be calculated that the vertical load to be supported by each bolt is 42 metric tons. The bolts are angled at 45°, so a load of 59 tons has to be applied to produce a vertical component of 42 tons.

Freysinnet high-tensile steel cable was used. Each cable assembly consists of a number of tendons, 15 mm in diameter each of which consists of 7 twisted strands of wire. The ultimate strength of each tendon is 26 tons and usually they are tensioned to 80 per cent of this load or 21 tons. Hence, three tendons per cable bolt would provide an applied load of 63 tons and an ultimate load capacity of 79 tons. This was considered sufficient to support a roof section 2.4 m thick.

Two hundred and fifty of these cable bolts were grouted in place and tensioned. Figure 41 shows the load that was recorded on six of these cable bolts. After installatio the load relaxes due to the creep characteristics of salt. But after a short period the load stabilizes and increases slightly due to the mining activity connected with the mill construction. A year and a half after installation of the first cable bolt the whole system appears to be functioning as it was designed.

Cut-and-fill mining

Cut-and-fill is a major underground mining method in Canada and is extensively used in the nickel mines in the Sudbury Basin. Figure 42 shows a typical longitudinal section through a block of transverse cut-and-fill stopes and rib pillars. The stopes are 6 to 12 m wide and can extend to 120 m between hangingwall and footwall. The intervening rib pillars are usually 6 m wide. Mining commences at the bottom of the stopes and progresses upwards in a series of slices with subsequent backfilling. The backfill provides a working floor and restrains the pillars, hangingwall and footwall. After the stopes are completed the crown and rib pillars are recovered in the reverse sequence usin underhand cut-and-fill. A wooden mat is constructed after every slice to provide support for the fill while mining the next slice underneath.

One major disadvantage of this mining method is the pillar recovery operation, which is more expensive than primary stoping and it takes a relatively long time to complete. In some mines, this problem has been overcome by a new mining method, called the "post" pillar, developed by Falconbridge Nickel Mines Ltd. (41). Figure 43 shows an isometric sketch of the mining layout; basically it combines room-and-pillar and cut-and-fill mining. The very small "post" pillars extend vertically between footwall and hangingwall and are permanently left in place. The stopes are mined by normal overhand cut-and-fill techniques. The rock mechanics aspect of this mining system is the determination of the size of the very slender "post" pillars.

A previous example (5.3) showed the method of designing rigid pillars. These pillars have a safety factor of greater than one and are designed to support the total weight of the overburden. In the "post" pillar mining method the pillars are designed to yield and they have a safety factor of less than one. The section on rock mass strength showed tha a pillar, after passing through its peak strength, can still retain considerable support capacity even though it is extensively fractured (figure 16). For successful development of the "post" pillar technique it is important that the pillars gradually fall within the fill and prevent sudden collapse. The fill prevents complete disintegration of the pillars and additional cement can be added for more rigidity, if required.

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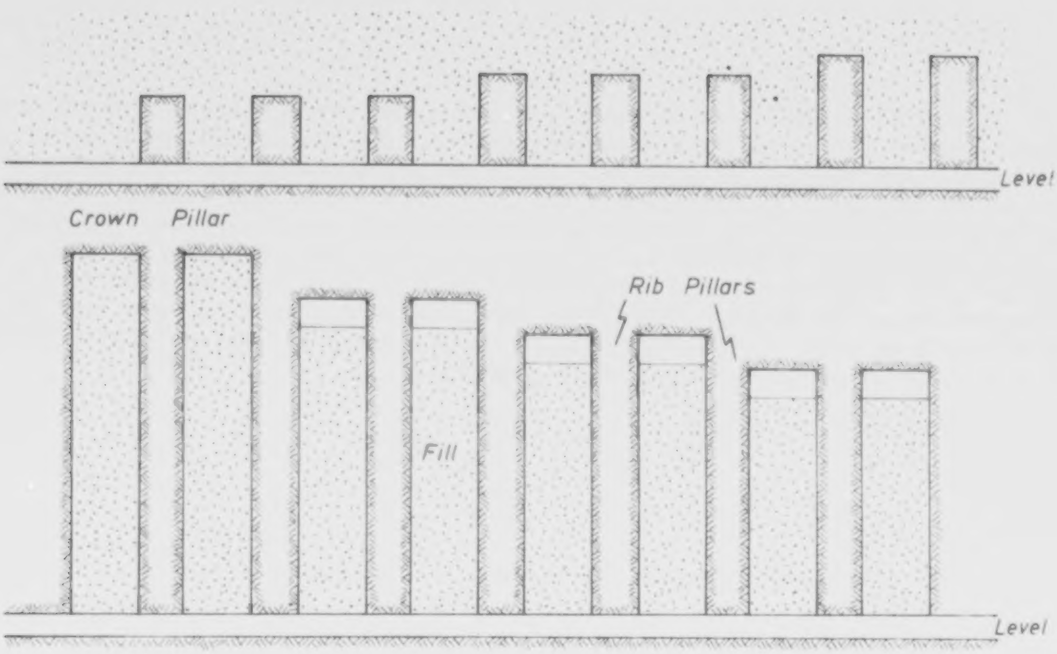
Level
Crown Pillar
Rib Pillars
Fill
Level

Figure 42. Layout of Transverse Cut-and-Fill Stopes

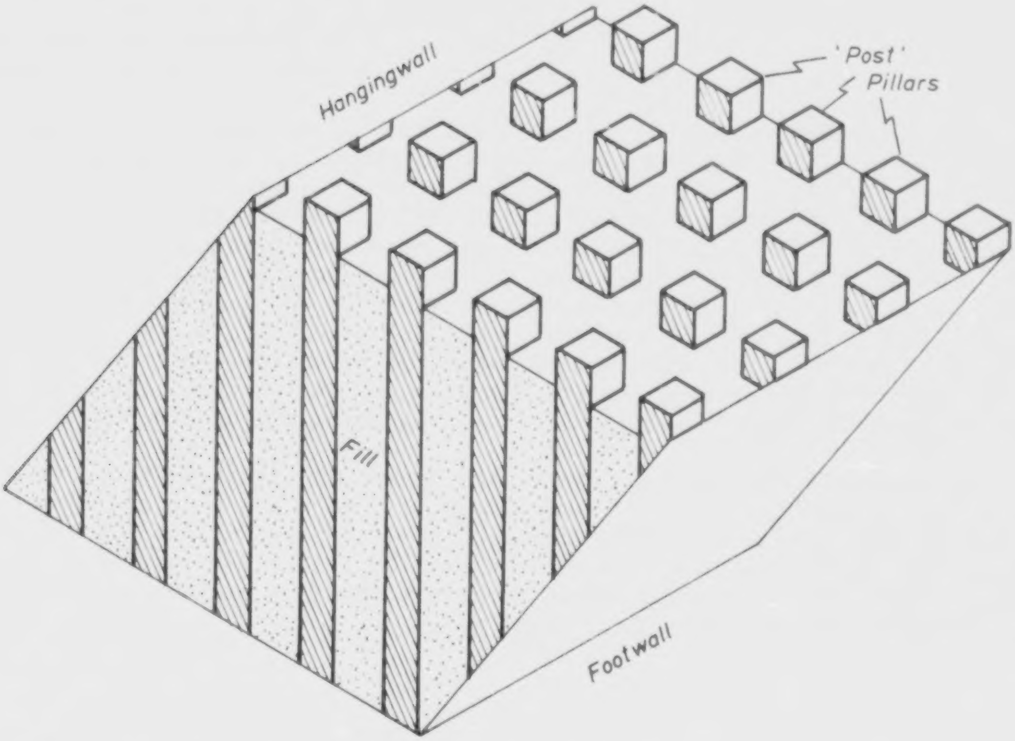
Hangingwall
'Post' Pillars
Fill
Footwall

Figure 43. Layout of "Post" Policy Mining Method

Figure 44 shows the relationship between the various factors which determine the dimensions of the "post" pillar mining method. In the lower-left quadrant, pillar and room width are related to give extraction ratio, assuming that the pillars are square and all rooms are the same width. The lower-right quadrant relates extraction ratio and depth to pillar stress, assuming that the full weight of the overburden is supported by the pillars and that the vertical stresses increases 0.25 kg/sq cm per metre in depth In the upper-right quadrant, pillar stress and strength are related to the safety factor To ensure that the pillars yield they are usually designed to have a safety factor between 0.8 to 1.0. The most arbitrary relationship is between pillar width, height and strength, as shown in the upper-left quadrant:

$$S_p = S_o \left(0.75 + \frac{w}{5h}\right) ,$$

where,

S_p = pillar strength, kg/sq cm,

S_o = strength of a 1 m cube, kg/sq cm,

w = pillar width m,

h = pillar height m.

For very slender pillars, width and height have little effect on the strength, as illustrated by the near-horizontal lines in the upper-left quadrant. The strength of th 1 metre cube is the major factor, and this has to be estimated from compressive strength tests on both sound and fractured specimens, similar to that shown in figure 27.

The nomograph (figure 44) is used as a tool in conjunction with experience to determine what pillar and room dimensions are feasible. Two examples are illustrated in the nomograph. For a 5-m pillar width and a pillar height of 40 m, the pillar strength is 1,085 kg/sq cm. Using a safety factor of 0.8, the pillar stress is 1,355 kg/sq cm and at a depth of 750 m the extraction will be 86 per cent and the room width 8 m. At this point, it may be decided that the pillars are too small considering the overbrea which can be expected from blasting. Going through the same procedure for a 10-m wide pillar slightly increases the pillar strength, stress, and extraction and gives a room width of 17 m. This span may be considered too large for effective roof control. Hence at this point it can be estimated that a pillar width of about 7 m and a corresponding room width of 12 m will be acceptable with regard to pillar and roof stability.

The experience at Falconbridge Nickel Mines Ltd. (41) was that, at a depth of about 700 m, pillar widths of 6 m and room widths of 9 m on one side and 12 m on the other sid of the pillars, giving an extraction of 87 per cent gave acceptable mining conditions. It was considered important that measurements of pillar deformation be taken during the initial stages of this mining method to ensure that the pillars neither burst, owing to high stresses, nor are completely crushed.

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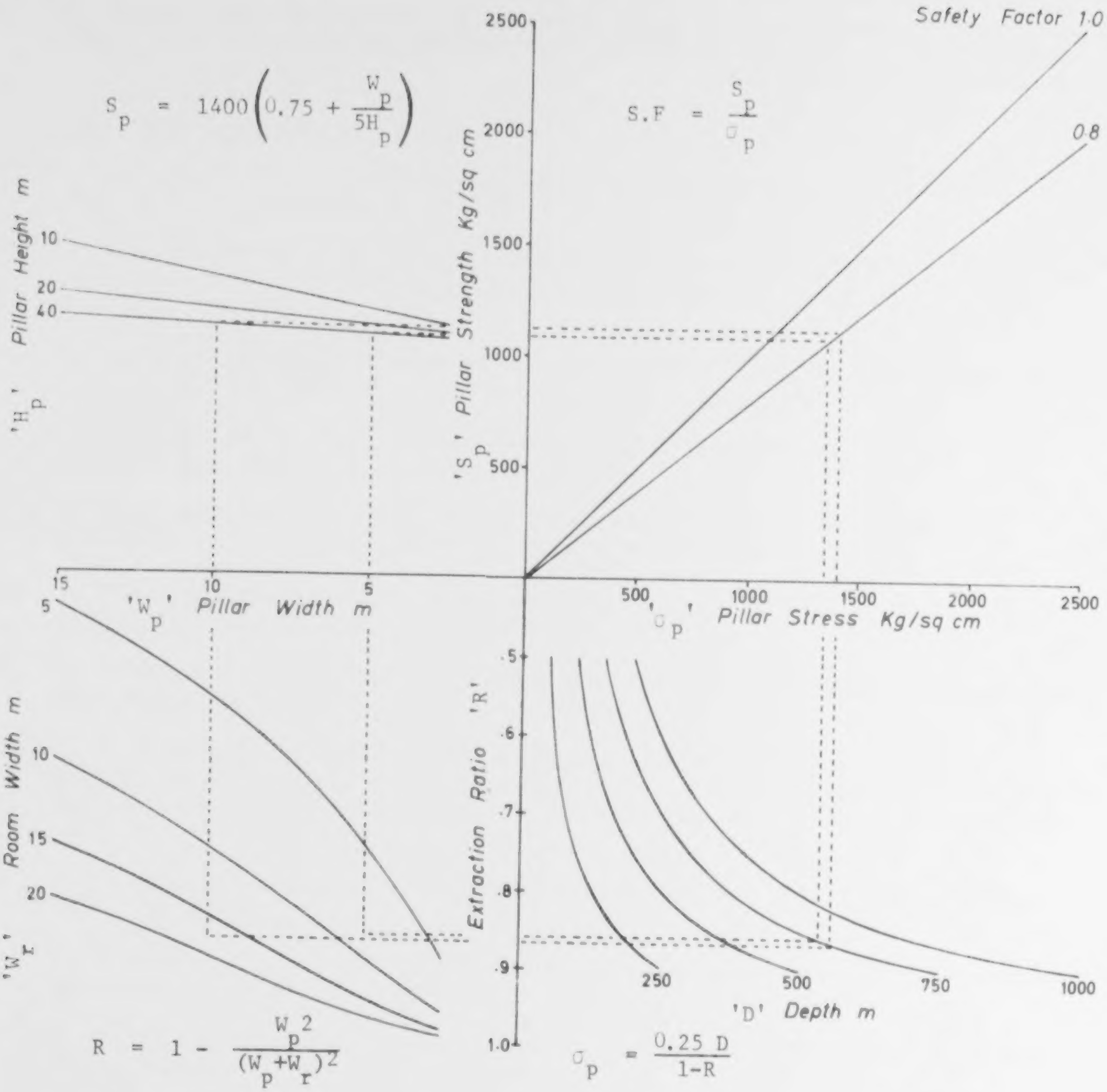

Figure 44. Design Nomograph for "Post" Pillars

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The design of mine structures is dependent on a number of interrelated factors, including transportation, ventilation, mining equipment and ground conditions. The purpose of the rock mechanics engineer is to provide relevant information to assist the person responsible for making decisions. This series of examples shows what has been achieved by some rock mechanics investigations and there are other examples available in the literature.

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V. MINE SUPPORT

J. C. Wilson*

This paper on mine support is presented, not from a theoretical viewpoint, but with strong emphasis on the practical applications of non-geological support media employed daily in the mines of our Sudbury operations where nickel-bearing ground is extracted safely and profitably.

The Sudbury operations of Falconbridge Nickel Mines Ltd. presently (1973) comprise eight producing mines, four concentrators and a smelter, within the world-renowned nickel-producing area known as the Sudbury Basin (figure 45). From a modest beginning in 1930 when 75,859 tons were hoisted to a 1972 tonnage of 4,190,980, many improvements in both practices and procedures involved in mine support have taken place. These have been brought about by necessity in that mining is now undertaken at greater depths where associated high rock stresses must be efficiently and effectively dealt with and, of course this must be achieved in an economic manner.

This paper deals briefly with the need for support in our underground operations and discusses, in more detail, the current support techniques, material, application and costs.

A. Requirement for mine support

The basic objective of mine support is to provide a safe and economic means of maintaining a man-made excavation over the required life span of the opening.

There are many interrelated factors which have a bearing, both directly and indirectly, on achieving this objective and, in turn, the selection of the support media to be adopted.

Of these factors there are two of overriding importance, namely geological and economical, which when analysed together determine the method of mining.

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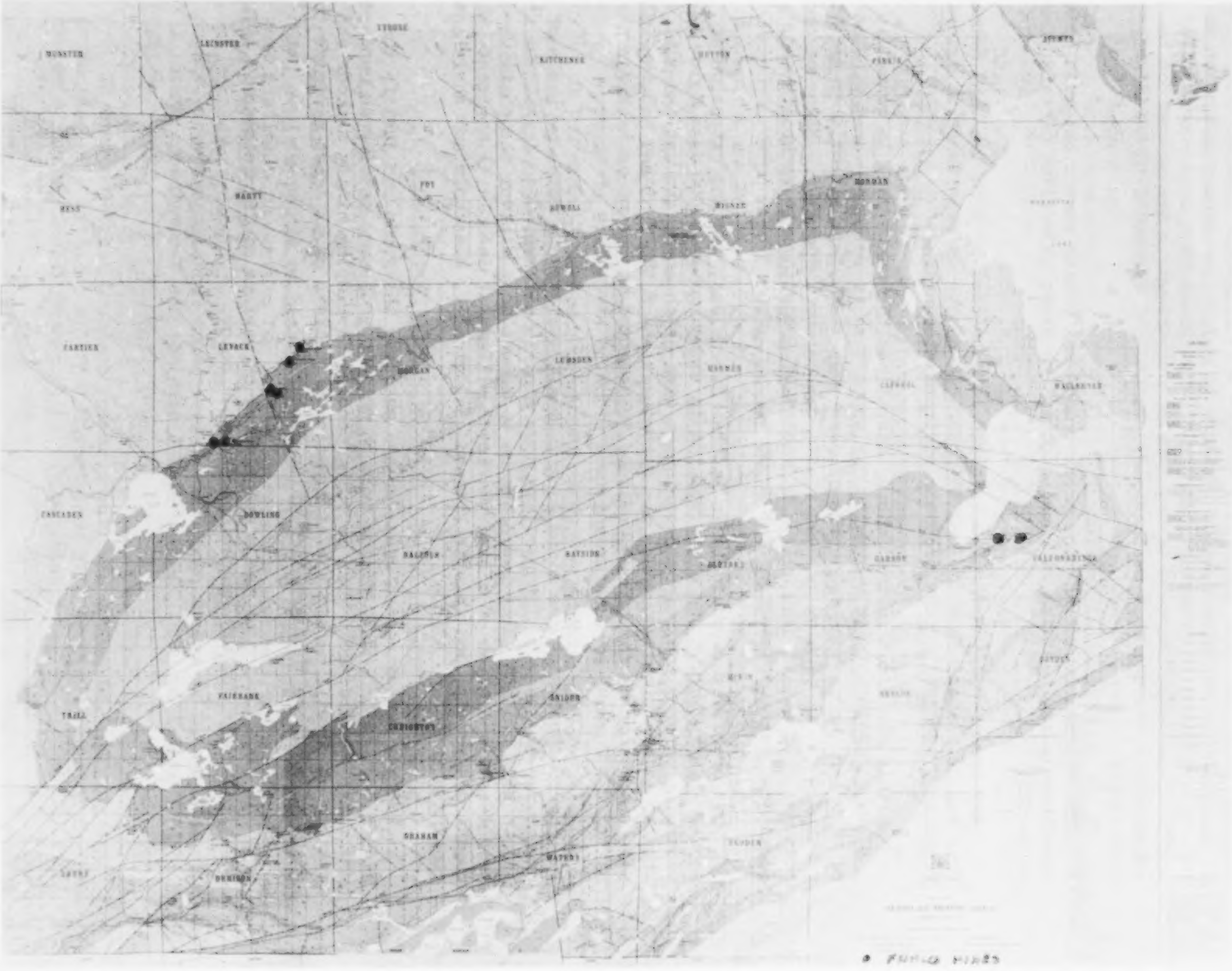
Figure 45. Geological map of Sudbury Basin showing operating mines


MUNSTER
LEINSTER
TYRONE
KITCHENER
HUTTON
PARKIN
HESS
HARTY
FOY
BOWELL
WISNER
NORMAN
CARTIER
LEVACK
MORGAN
LUMSDEN
HANMER
CAPREOL
CASCADEN
DOWLING
BALFOUR
RAYSIDE
BLEZARD
GARSON
FALCONBRIDGE
TRILL
FAIRBANK
CREIGHTON
SNIDER
GRAHAM
WATERS
DENISON
LORNE
NEELON
DRYDEN
• PROD. MINES

Another prime consideration is the role played by mine planning. The over-all mining geometry, long and short-term, from the viewpoint of support must be compatible with the grade, tonnage and economical cost of extraction at any point in the mine's life. Failure to pay the necessary attention to this has resulted in premature closure of what could have been economically viable operations. Support in the form of pillars which may or may not be subsequently removed must be carefully planned and sequenced to maximize profit with safety.

The optimum amount of mine support does not remain static but must be continuously reviewed in order to maintain the basic objectives of safety and economics.

Any one orebody during its life can be extracted by more than one method. This can be planned from the beginning or can evolve as experience is gained, technology advances or the economic environment changes.

Within our mines in the Sudbury operations the waste rock is generally competent requiring only rockbolts for support as is evidenced by square and rectangular shafts sunk to depths of just over 4,000 feet with timber sets used primarily for supporting guides, pipes, cables and ladderways. Miles of crosscuts and drifts have been driven in waste and ore with only rockbolts and occasional areas of guniting for support.

The orebodies vary widely in lateral dimensions, dip and extension in depth. Presently, at one mine, stoping is under way at 6,000 feet below surface. Generally, ground conditions deteriorate with depth and tight cut-and-fill stoping is practised to reduce the deterioration of the foot and hanging walls and remove some of the ground pressure transmitted to the stope back thereby making the working area safer and less prone to stope back problems.

Narrow orebodies are mined longitudinally, wider orebodies by means of transverse stopes with intermediate down dip rib pillars. Stoping blocks are laid out in two and three level blocks to reduce to a minimum the costly and troublesome operation of mining the level pillars at each level horizon. Wherever conditions permit, and this is limited geologically, blasthole and shrinkage stoping methods are adopted.

Pillars are extracted whenever and wherever the grade can justify the higher cost of secondary mining by such methods as square setting and undercut and fill. Considerable work is being done on developing mining methods and systems which will permit the elimination of pillars or at least minimize the quantities of ore that are tied up in pillars.

The distribution of the tonnage mined by various mining methods in 1972 at Falconbridge Sudbury operations is shown below:

Mining Method	1972 tonnage (percentages)	
Transverse cut-and-fill	2.9	primary fill
Longitudinal cut-and-fill	29.4	
Undercut-and-fill	6.2	
Mechanized cut-and-fill (Strathcona mine)	48.7	
Square set	3.1	
Shrinkage	1.0	delayed fill
Blasthole	7.1	
Development	1.6	

The role of tailing fill as a support medium, called upon by the choice of mining method, is well illustrated by the above table.

Extremely bad ground conditions in some ore zones coupled with high lateral wall pressures which are frequently encountered, at depth, when a stope approaches within 30 ft of a level requires the adoption of close timbering methods, such as square setting, to provide a safe method of support. This use of timber is one of the few remaining applications of timber in our U/G operation for the direct support of mine openings.

Indirectly timber is still extensively used to maintain openings for service purposes such as gangways within the active mining areas of the orebody. Steel in the form of yielding arches is being used for similar purposes where its longer life has economic advantages.

In our underground mining operations the requirements for mine support for the safe and economic extraction of sulphide ores is an integral part of the entire mining procedure. It is a prime consideration in all phases from initial planning through to sealing of the shafts after the last ore has been hoisted.

B. Support media: characteristics, applications and costs

1. Hydraulically placed tailing and cement-consolidated tailing fill

In the early days of cut-and-fill mining, particularly at Falconbridge mine, ample supplies of gravel and sand overlying and adjacent to the orebody were available on surface for filling purposes. The material was introduced into the mine in a dry state and distributed by mechanical means. This material was considered unsatisfactory as time went by because of the depletion of supply, increasing labour costs, relatively poor support of the material because it lacked cohesiveness and the high labour content inherent in the distribution system.

To overcome these problems research work was conducted, not only into the production of a suitable material direct from mill tailing, but also in the system of hydraulically transporting the material U/G.

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By the late 1940s the use of classified mill tailing provided our mines with a substantially cheaper and more effective method of back filling and supporting mining excavations. Despite this advance certain problems remained such as the inability of tailing to consolidate, shrinkage in volume after emplacement, the necessity of providing timber in transverse stoping to construct pillar fences to contain the fill when extracting pillars and the need for high cost, both in labour and materials, for timber plank floors to avoid dilution in primary mining. To overcome the latter problem the technique of consolidating tailing with cement was developed and successfully introduced in the late 1950s. A 6" thick floor of cement-tailing, in the ratio of 1:6, having a compressive strength of 400-500 psi after 28 days was placed hydraulically over the previously poured tailing. Following this use of cement came the development of bulk consolidation of tailing with cement which has now become a recognized practice. This development has overcome most of the previously mentioned inherent drawbacks of tailing.

Material

Classified mill tailings constitute the bulk of back fill material. Development waste wherever possible is dumped as fill and screened alluvial sand is added to the tailing fill preparation circuits in the ratio of 1:1 to augment deficiencies in the tailing supply as they occur.

Typical screen analyses are as follows:

(a) Mill and mine tailing:

Mesh	Mill tailing or classifier feed		Classifier overflow		Classifier underflow or mine tailing	
	Wt %	Cum %	Wt %	Cum %	Wt %	Cum %
+ 35	0.4	0.4	-		0.8	0.8
+ 48	2.3	2.7	-		4.3	5.1
+ 65	6.4	9.1	0.2	0.2	11.4	16.5
+100	11.8	20.9	0.3	0.5	18.8	35.3
+150	10.0	30.9	0.4	0.9	15.5	50.8
+200	11.0	41.9	0.6	1.5	14.8	65.6
+270	6.8	48.7	0.7	2.2	9.0	74.6
+325	6.6	55.3	1.4	3.6	7.8	82.4
-325	44.7	100.00	96.4	100.00	17.6	100.00

(b) Screened alluvial sand:

Mesh	% Wgt	Cum %
+ 10	18.8	18.8
+ 14	10.0	28.8
+ 20	9.9	38.7
+ 28	12.8	51.5
+ 35	15.6	67.1
+ 48	12.9	80.0
+ 65	8.3	88.3

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Mesh	% Wgt	Cum %
+100	6.6	94.9
+150	2.3	97.2
+200	1.4	98.6
+270	0.6	99.2
+325	0.2	99.4
-325	0.6	100.0
	100.0	

The percolation rate for the above sample was 69.9"/hr.

(c) Normal Portland cement:

Portland cement of normal fineness, blaine 3700 sq cm/gm, with 96.3 per cent passing 200 mesh and 89.0 per cent passing 325 mesh is the common type of cement used.

Chemical analysis of tailing fill from two mines, one on each side of the Sudbury basin show the following composition (percentages):

	Falconbridge mine	Fecunis mine
Silica	47.48	36.62
Alumina	14.66	24.28
Iron oxide	15.92	22.22
Lime	6.23	5.04
Magnesia	6.11	3.64
Sulphide sulphur	(not determined)	3.99
Sulphur trioxide	0.59	0.38
Loss on ignition	4.70	2.82

Preparation

A fill preparation plant should be designed to provide the mine with a quality product in sufficient quantities to satisfy demand at the lowest possible cost.

In our plants mill tailing is deslimed by means of cyclones to give an optimum particle size distribution consistent with cost and maximum utilization of fill material resulting in a tailing having a percolation rate of 4-5"/hr. This product is placed in agitated storage tanks for immediate use or in settled storage from which it is drawn as required. Cement circuits are an integral part of the plant design with automated cement addition to minimize labour content and over-all cost of cemented fill. The control of fill density, mixing time and correct proportioning of cement are of the utmost importance to ensure optimum strength at minimum cost. In large plants where close to 600 t/hr are poured through three lines to U/G at any one time, the instrumentation to achieve this control is costly and complex (figure 46).

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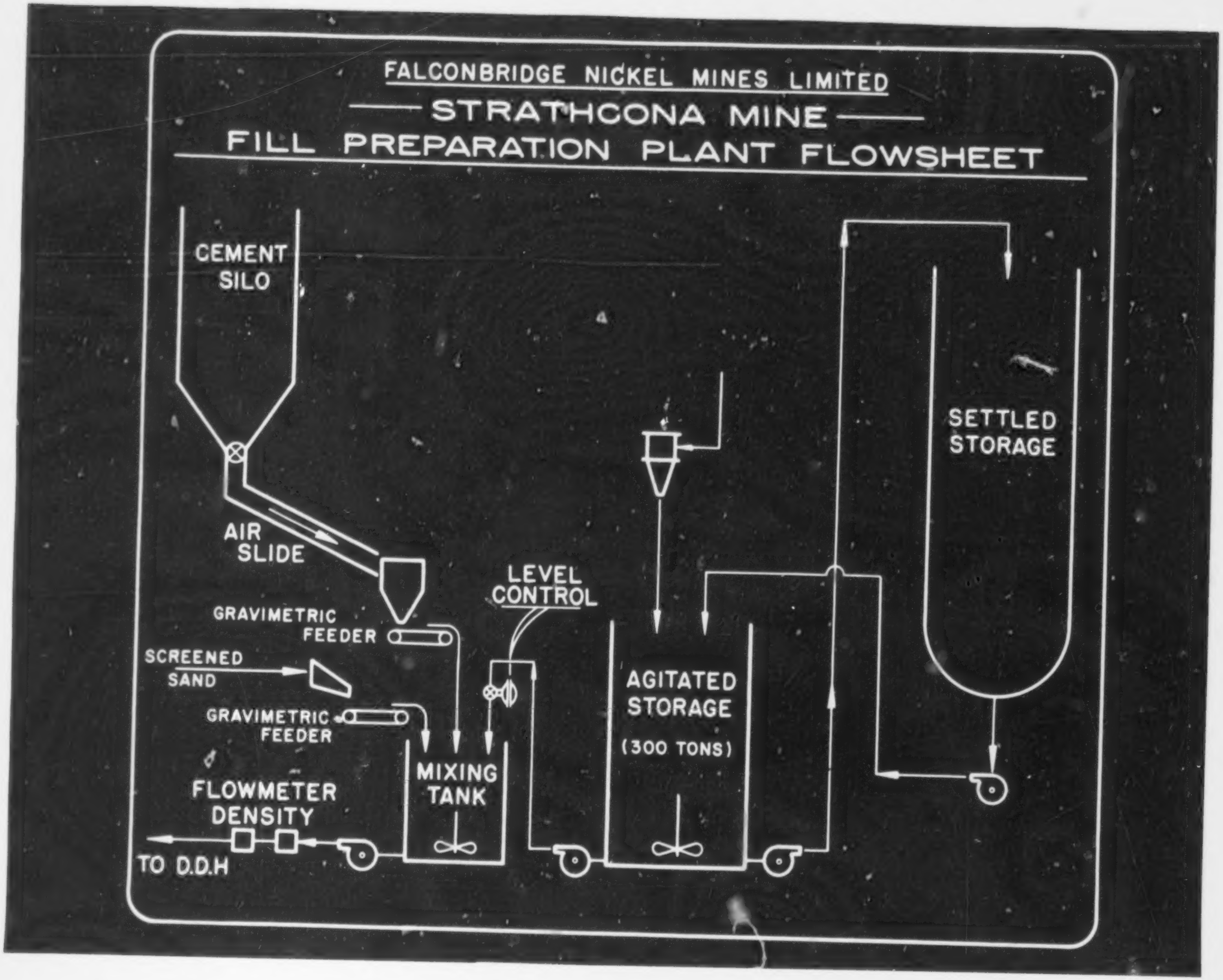
Figure 46. Strathcona mine: fill preparation, plant flowsheet


FALCONBRIDGE NICKEL MINES LIMITED
STRATHCONA MINE
FILL PREPARATION PLANT FLOWSHEET
CEMENT SILO
AIR SLIDE
GRAVIMETRIC FEEDER
SCREENED SAND
GRAVIMETRIC FEEDER
FLOWMETER
DENSITY
TO D.D.H
LEVEL CONTROL
MIXING TANK
AGITATED STORAGE
(300 TONS)
SETTLED STORAGE

At all plants cement bulk storage silos are provided varying in capacity from 35 tons to 450 tons. The cement is delivered in 25-ton bulk transporters and unloaded pneumatically into these silos.

Application

In 1972 our Sudbury mines hoisted 4,190,980 tons of which 87 per cent came from cut-and-fill stoping. The importance and increasing role played in our operations by tailing and cement consolidated tailings is shown by the following table:

Year	Cement	(tons)	Tailing fill
1960	719		802,187
1961	775		1,078,706
1962	1,387		1,076,032
1963	2,113		910,741
1964	8,402		974,996
1965	19,008		1,126,123
1966	16,870		1,068,429
1967	25,510		1,030,951
1968	39,932		1,139,228
1969	35,932		1,186,470
1970	71,598		1,892,675
1971	77,343		1,983,949
1972	62,401		1,953,021

The use of hydraulically placed tailing and particularly cement consolidated tailing has made it possible, through the supporting characteristics of these forms of backfill, to economically mine to depths of 6,000 ft by cut-and-fill method.

A further important factor in the use of cement in tailing fill is that it has made possible a higher percentage recovery of ore by relatively less expensive primary mining methods thereby minimizing or eliminating pillar mining. It has also lowered the economic cut-off grade of pillars scheduled for subsequent mining.

Some specific applications of a higher strength cement consolidated tailings are:

Mucking floors. For all mines approximately 50 per cent of our U/G cement consumption is used for cement content mucking floors. Typical mixtures, strengths and thicknesses are:

Cement-tailing ratio	28-day strength (psi)	Thickness
1:6	400-500	6"
1:8	200-300	12" for LHD equipment
1:12		12" for scraping

/...

To what degree these relatively thin bands of "high" strength material, placed in some instances on unconsolidated fill, provide support to the adjacent ground is not known; however, when it is considered that these bands occur at vertical intervals of 9 ft throughout the filled stope, their supporting effect, en masse, particularly in narrow stoping widths, must be of some appreciable magnitude.

Bulk pours. These pours, with cement:tailing ratios of 1:16, 1:30 and 1:32 have been placed underground to provide a free standing structurally self-supporting mass which eliminates timber fill, retaining fences adjacent to pillars scheduled for subsequent recovery (figure 47).

In transverse cut-and-fill mining where primary stopes have been filled with 1:30 mixes and without pillar fences, the subsequent extraction of the pillars, either by square set or UC and F methods, has benefited in that the free standing fill does not slough, thereby minimizing the dilution of ore from fill runs and in addition it has provided local support to the fractured pillar surfaces.

One lenticular orebody dipping at between 30-35° and having a maximum length of 700 ft and thickness of 150 ft was divided up into 25-ft wide transverse stopes and 50-ft wide pillars with longitudinal stopes at the strike extremities. The transverse stopes were mined overhand cut-and-fill and filled with 1:30 cemented fill and 15-ft wide blocks of consolidated fill were poured adjacent to the pillars in each longitudinal stope. No timber fill fences were installed (figure 48). These massive pours of cemented fill provided sufficient support to the flat hanging wall so that the 50-ft wide pillars were successfully and economically mined 60 per cent by open cut-and-fill and 40 per cent by UC and F methods.

At one mine three 60-ft wide transverse blasthole stopes were filled prior to mining the two intervening 60-ft wide transverse pillars. The west stope was filled with tailing, the centre stope with 1:16 cement:tailing and the east stope with 1:32 mix. Presently the western pillar is being extracted by a modified blasthole retreat method starting from the top down, leaving a nominal 10-ft skin of ore adjacent to the tailing fill west stope and breaking to the 1:16 mix in the centre stope. To date approximately 40 ft of the 1:16 mix has been exposed for a length of 200 ft and no sloughing observed.

A major factor contributing to the successful application of UC and F mining is the self-supporting characteristic of cemented tailing in that the dead weight on supporting timber has been considerably reduced and appreciable economies in the use of timber effected (figure 49).

In the recently developed post pillar mining method at our Strathcona mine 1:32 cement:tailing mix is placed as fill (figure 50). The initial function of the fill poured around a 20 x 20 foot post pillar is to restrain the fractured rock and thereby significantly increasing the pillar strength. Another added advantage from this cement-consolidated fill is the higher density or lower porosity which reduces the compaction and improves the load bearing capability of the fill. The migration of fine cement particles into open fractures within the pillar also improves the pillar strength.

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Figure 47. Photo of free-standing cement: tailing without fill fences

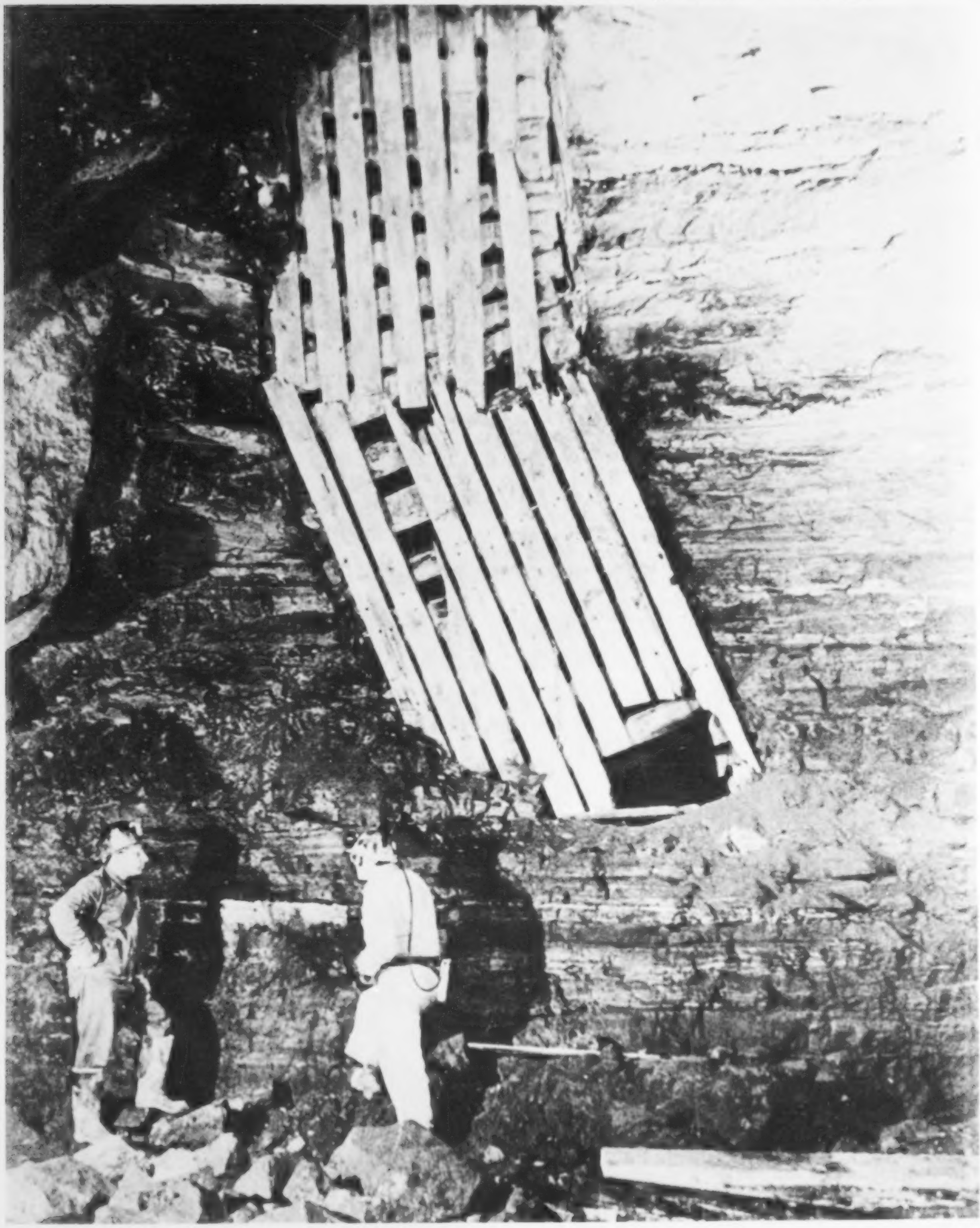


Figure 48

PLAN OF ONAPING WEST OREBODY BELOW 1650 L.

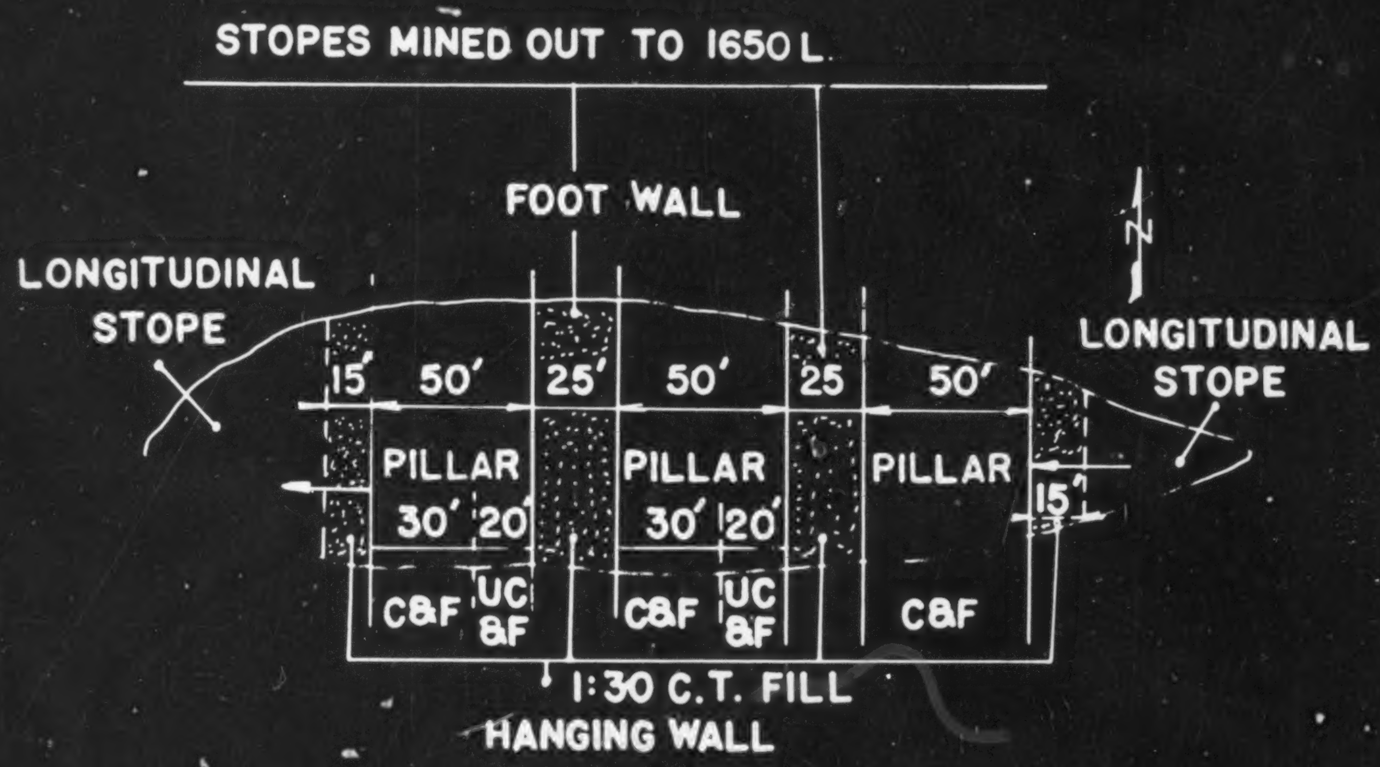

STOPES MINED OUT TO 1650 L.
FOOT WALL
LONGITUDINAL STOPE
LONGITUDINAL STOPE
N
15'
50'
25'
50'
25
50'
PILLAR
PILLAR
PILLAR
30'
20'
30'
20'
15'
C&F
UC &F
C&F
UC &F
C&F
1:30 C.T. FILL
HANGING WALL

FIG. 4.

Figure 49. Undercut and fill, showing timber support

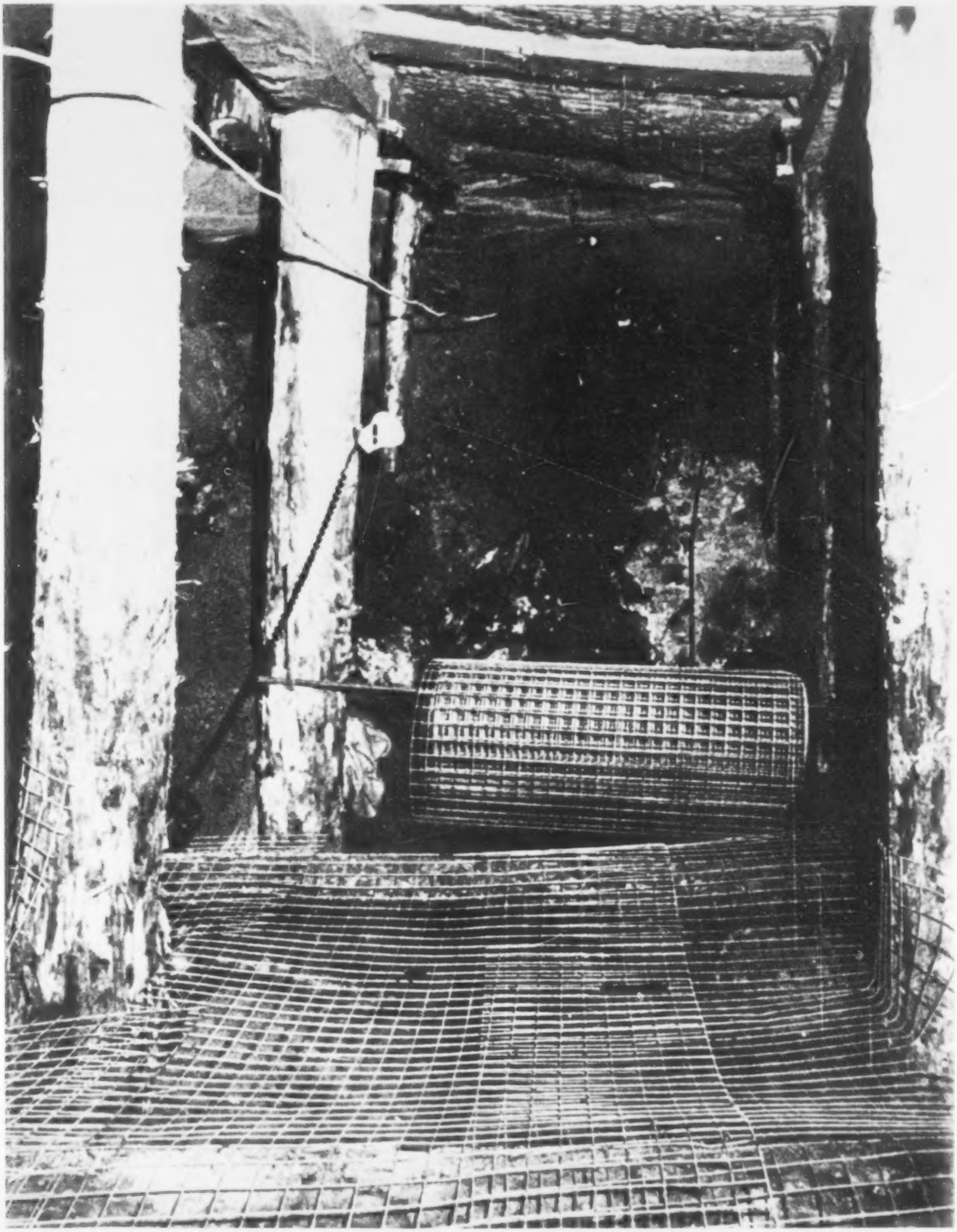
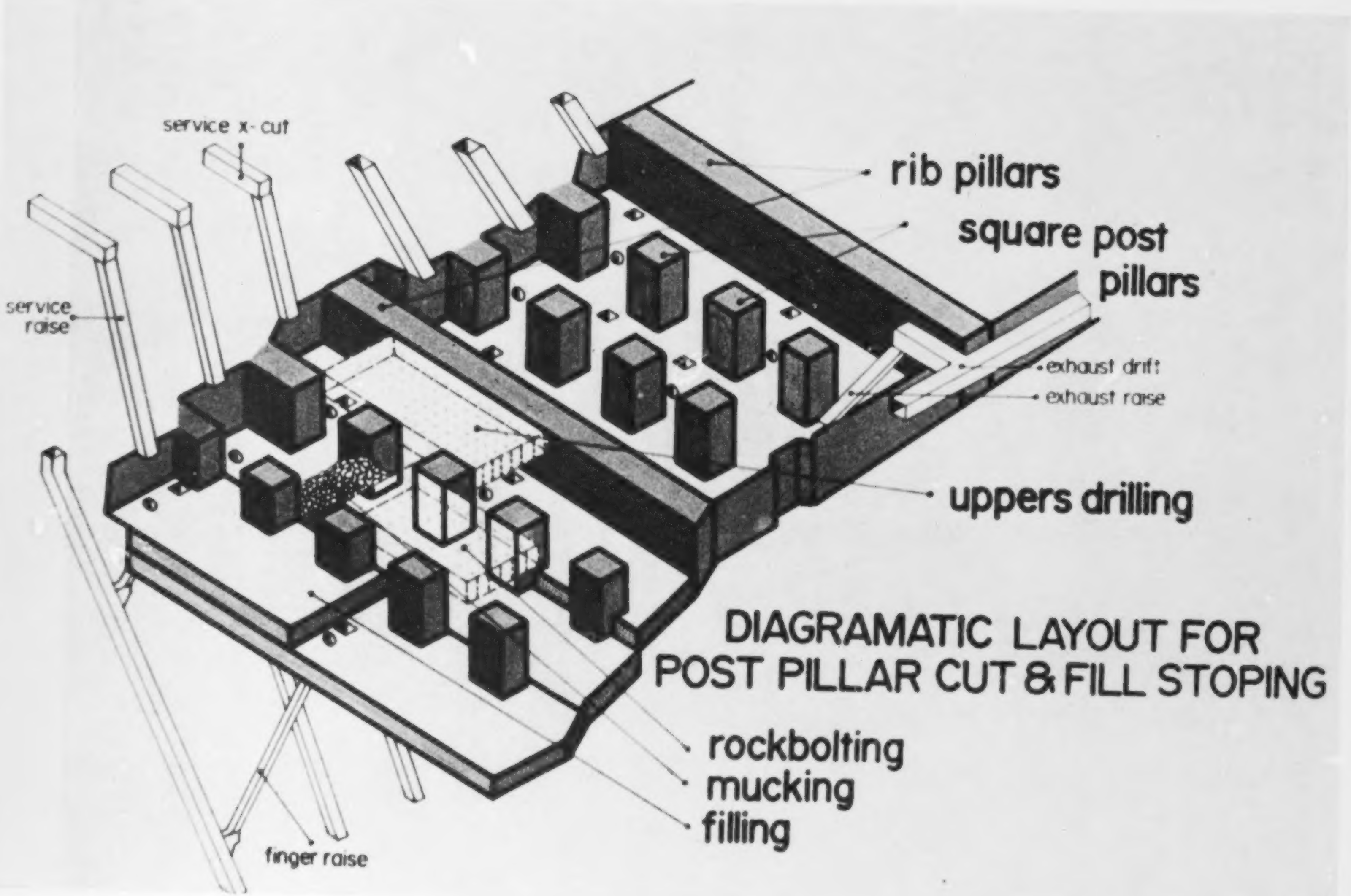


Figure 50


rib pillars
square post pillars
exhaust drift
exhaust raise
uppers drilling
DIAGRAMATIC LAYOUT FOR POST PILLAR CUT & FILL STOPING
rockbolting
mucking
filling
service x-cut
service raise
finger raise

Typical 28 day strengths of 1:20 and 1:30 cement:tailing mixes are 70-100 psi and 40-70 psi respectively.

Costs

The cost of operating the tailing fill plants, transporting the tailing underground and preparing and pouring in the stope varies, depending upon the design of the plant, the age and complexity of the pipeline-borehole transportation system and the different mine practices involved in placing the fill. An average cost is about $1 per 10 tons of fill in place. This figure does not include the cost of building fill fences and raising millholes or manways which are common for any form of fill and an integral part of the cut and fill mining method.

Bulk cement is delivered to our mines for approximately $30.00/ton. Because of the relatively high cost of this material every application must be analysed in detail to justify the cost of cement addition. Through experience the most economical mix must be established. Initial capital for cement addition facilities in the surface fill plant must also be considered.

Common mixes, approximate costs and typical applications are:

Mix (Cement:tailing)	Cost per ton in place	Application
1:6	$5.00	Scraping floors, 6" thick narrow stopes
1:8	$4.00	Mucking floors, 12" thick with LHD equipment
1:12	$3.00	Scraping floors, 12" thick wide stopes
1:30-1:32	$2.00	Bulk consolidation for ground support

2. Timber

All timber supplied to our underground operations is obtained from a wholly owned subsidiary company of Falconbridge Nickel Mines, Ltd., which, from its own local timber concessions, provides 70 per cent of our annual requirements. Of the remaining consumption 10 per cent is Douglas fir, obtained from the Canadian west coast, and 20 per cent is in the form of rough lumber and planking obtained locally from other sources.

Timber consumption has remained fairly constant over the last 11 years at a rate of 10 million board feet. However the mined tonnage has increased, with a subsequent reduction in timber/ton mined, reflected in the increasing use of rockbolts and changes in mining techniques which have reduced high-cost timber-intensive operations such as square setting. Post-pillar mining and the introduction of tailing fill and consolidated tailing fill, as previously described, are further examples of timber-reducing techniques.

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Material

Suitable mine timber for ground support purposes is available within Canada, but unfortunately at an increasing cost of about 10 per cent annually. The following list shows the principal types of timber, form and sizes and common U/G usage.

Type	Form and size	Treated	Common usage
Red pine and Jack pine and Spruce (Eastern species)	Stull posts 10" tops 10' long Bark off	Yes	Gangway posts
	Lagging 4-6" tops, 10'-16' long	No	Minor support, fill fences
	Bark on		Gangway lagging
	Bark off	Yes	
	Planks		
	3" x 8", 6'-16' long	Yes(50%)	Gangway lagging
	4" x 12" (pines only)	Yes	(Sill floors, gangway (cushion blocks
	2" x 12' (spruce only)	No	Laminated beams UC&F
	Square sets (80% pines) Posts, caps, girts, 8" flatted	Yes	Pillar extraction
	Framed timber 6", 7" flatted	Yes	Manway cribbing
Douglas fir	Square timber 12" x 12", 8'-20' long 10" x 10"	Yes	Gangway caps U/G construction Shafts sets
	Shaft guides 6 1/2" x 7 1/2"	Yes	

Two timber treatment processes are in use. About 50 per cent of all U/G timber is treated with chromated zinc chloride in a 10 per cent water solution which has proven effective up to 15 years as a rot preventative and fire retardant. The cost of this treatment is \$10-\$12/1000 board feet. Surface timber is treated with 5 per cent oil solution of penta-chlorophenol at a cost of \$25-\$30/1000 board feet, principally as a rot preventative, and has a proven effectiveness up to 20 years. A vacuum process is used to apply the chemicals to the timber surfaces.

Application

Generally speaking timber is used in our mines indirectly for ground support in that it is used to form a structure to maintain an opening such as level gangways and manways not intimately in contact with rock surfaces - a major exception being square set stoping.

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In development. Square and rectangular shafts have been sunk to depths of 4,422 ft using timber sets. The primary functions of the sets being to provide compartments within the shaft and a structure to which can be attached guides, pipes, electrical cable, manway stages and ladders. Little if any ground support to the shaft walls is provided by the sets.

The use of timber to support drift and x-cut openings in our mines is now very rare. It is only when highly fissured or blocky ground is intersected by these openings that pin sets or drift sets will be installed and blocked tight to the back.

Steeply inclined raises are bored or driven by means of Alimak raise climbers. No timber is placed for support but round lagging posts are subsequently installed to partition off the manway if the raise is to be used as a travelway.

Rock bolting and, if required, guniting have virtually replaced timber as a method of ground support in development openings.

Stope preparation. The practice at some of our mines is to extract the ore by longitudinal cut and fill stoping. The first stage in preparing the stope is to slash the walls of the level drift, which is driven in ore, and slash down the drift back. Timber gangway sets (figures 50, 51) are then installed at 3'-2" centres for the full length of the stope with provisions made for the construction of chutes at predetermined intervals. This operation consumes a major portion of our U/G timber requirements. Such timbered sets provide little direct support to the rock surfaces except in narrow ore widths where the 12" x 12" caps butt against the foot and hanging walls. Their main function is to maintain a service opening beneath the filled stope as it is progressively mined up dip from its level of origin.

Stoping. It is in square-set mining that timber is used directly as a method of ground support. In our mines this method of mining, because of its low productivity and high cost, is limited to the extraction of high grade and of badly fractured or highly stressed pillars.

Cap, girts, and posts are framed from 8" flatted pine timber. The sets are of the "cap butting" type, measuring 6 ft capwise, 6'4" gertwise and 8 ft high. Back blocking of 7" x 7' flatted pine or spruce is placed to transmit the greater portion of ground weight to the posts and to permit the erection of sets with a minimum of disturbance to blocking on the previous cut. Blocks from sets to walls are of 8" flatted material. If the distance from the centre line of the post to a wall exceeds 2 feet a narrow set is framed before blocking is wedged in place. Round lagging is used for face blocks between a set and the breast, since their use is temporary. After a set has been blocked and wedged where necessary it is covered with 3" planks to provide a safe headcover.

The UC and F method of pillar extraction also consumes a high proportion of our U/G timber requirements. Spruce planks 2" x 12" x 12' are used to construct six-piece laminated stringers which support a mat of round lagging beneath cement tailing fill. As these stringers are exposed by mining the ore beneath this mat, they are supported at 6-ft intervals by pine or spruce stull posts (figure 52).

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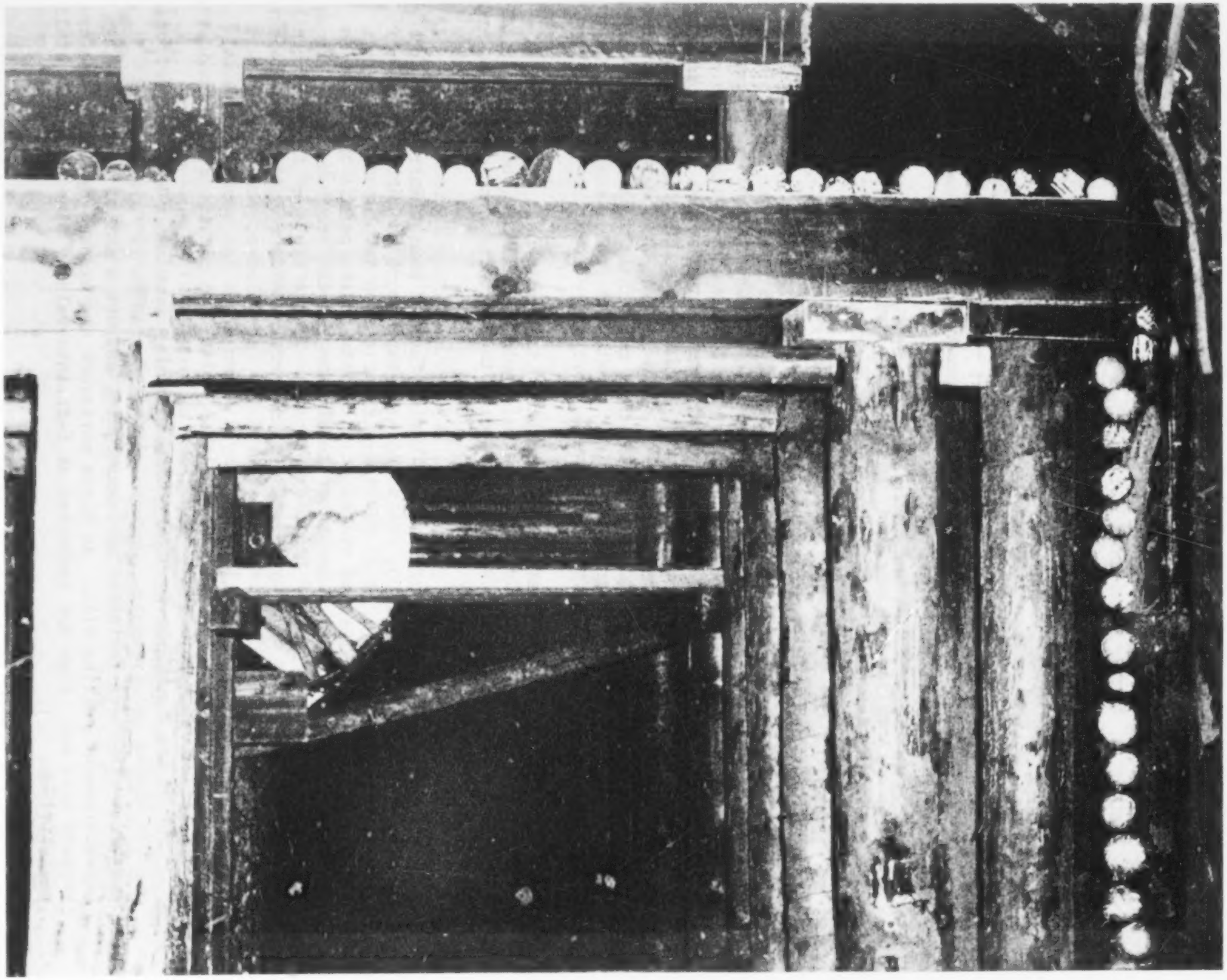
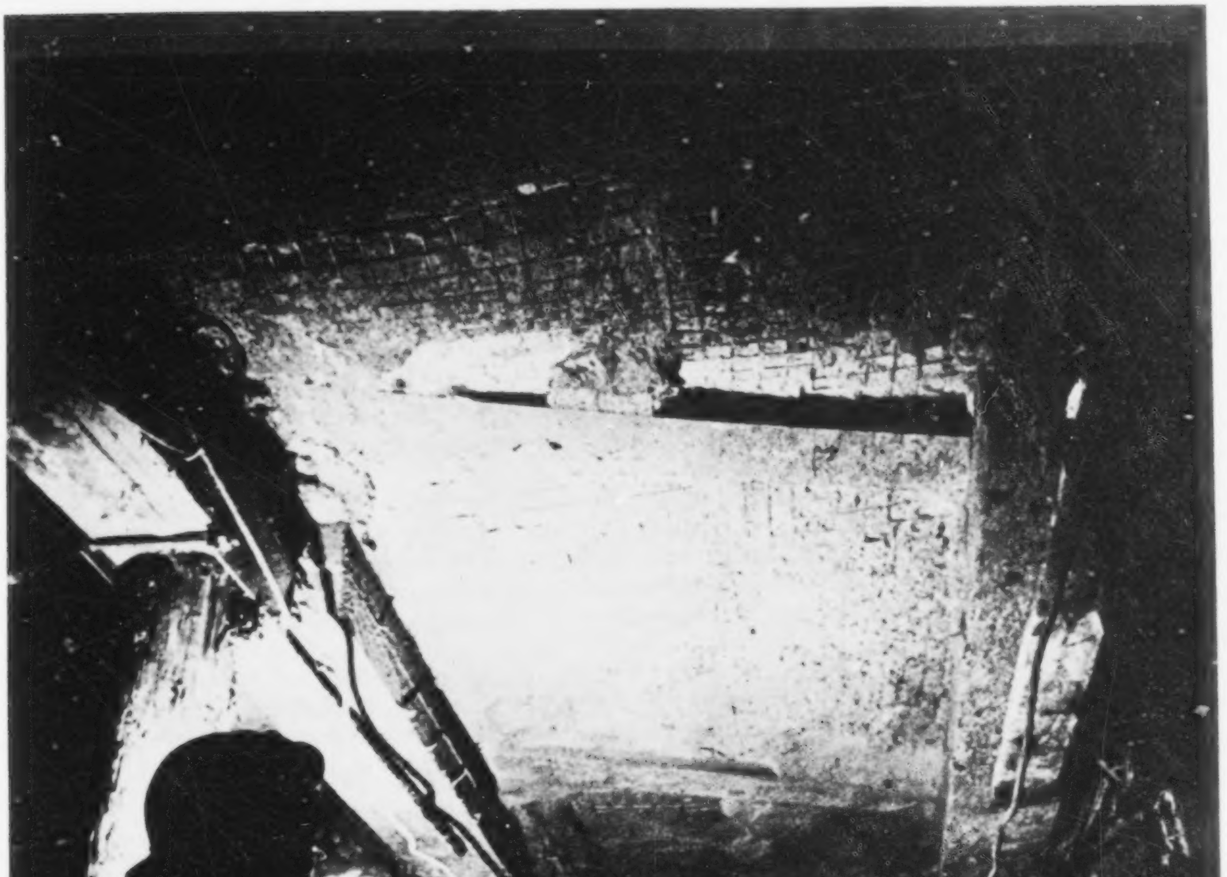


Figure 51. Gangway sets

Figure 52. Undercut and fill, showing timber



Again, this use of timber is not directly for ground support but to support the fill and to maintain the temporary opening, required for the removal of broken ore.

3. Steel

To date, steel has not been widely used in our mines for direct ground support.

It has found use in shafts lined with monolithic concrete or concrete rings to support guides and service lines, to replace timber support through badly broken faulted areas and also to replace timbered gangways in selected locations.

Rectangular rigid steel gangway sets consisting of a 17" x 7" wide flange or a 15" x 5 1/2" I beam cap placed 9'6" from top of rail and supported by 4" x 4" or 6" x 6" wide flange steel posts have been used.

Yielding three-piece steel arches weighing 16.8 and 14.1 lb/ft have been used successfully to support faulted ground (figure 53). Rigid and yielding two-piece light weight (8.6 lb/ft) steel arches have replaced timber gangway sets in locations where the expected service life of treated timber would be exceeded. The yielding arch is preferred to the rigid steel sets.

Experience to date with light weight yielding arches at 3-ft centres indicates potential for wider use as a replacement for timbered gangway sets. Despite their higher material cost the lower labour and maintenance cost provides a worth-while advantage over timber in this application.

4. Rockbolts

The support of mine openings, utilizing rockbolts, is a well established practice in all our mines. In 1972 a total of 457,000 bolts were used for long- and short-term support.

In our hard rock, bolts are used primarily to lock individual blocks into a self-supporting arch and to secure loose layers to a deeper stratum of solid rock. The principle of rockbolting to create a laminated beam spanning the opening is discounted in our type of ground.

Some advantages derived from this method of support are safety, reduction of mine timber, reduction in fire hazard, improved ventilation, increased mobility of mechanized equipment, reduction in surface and U/G storage space and the support of mesh, catwalks and pipes.

Rockbolts and accessories

Throughout the mines we have standardized on the expansion shell design bolt threaded at both ends and supplied in 5', 6' and 7'6" lengths for 1 1/4" diameter drill holes (figure 54).

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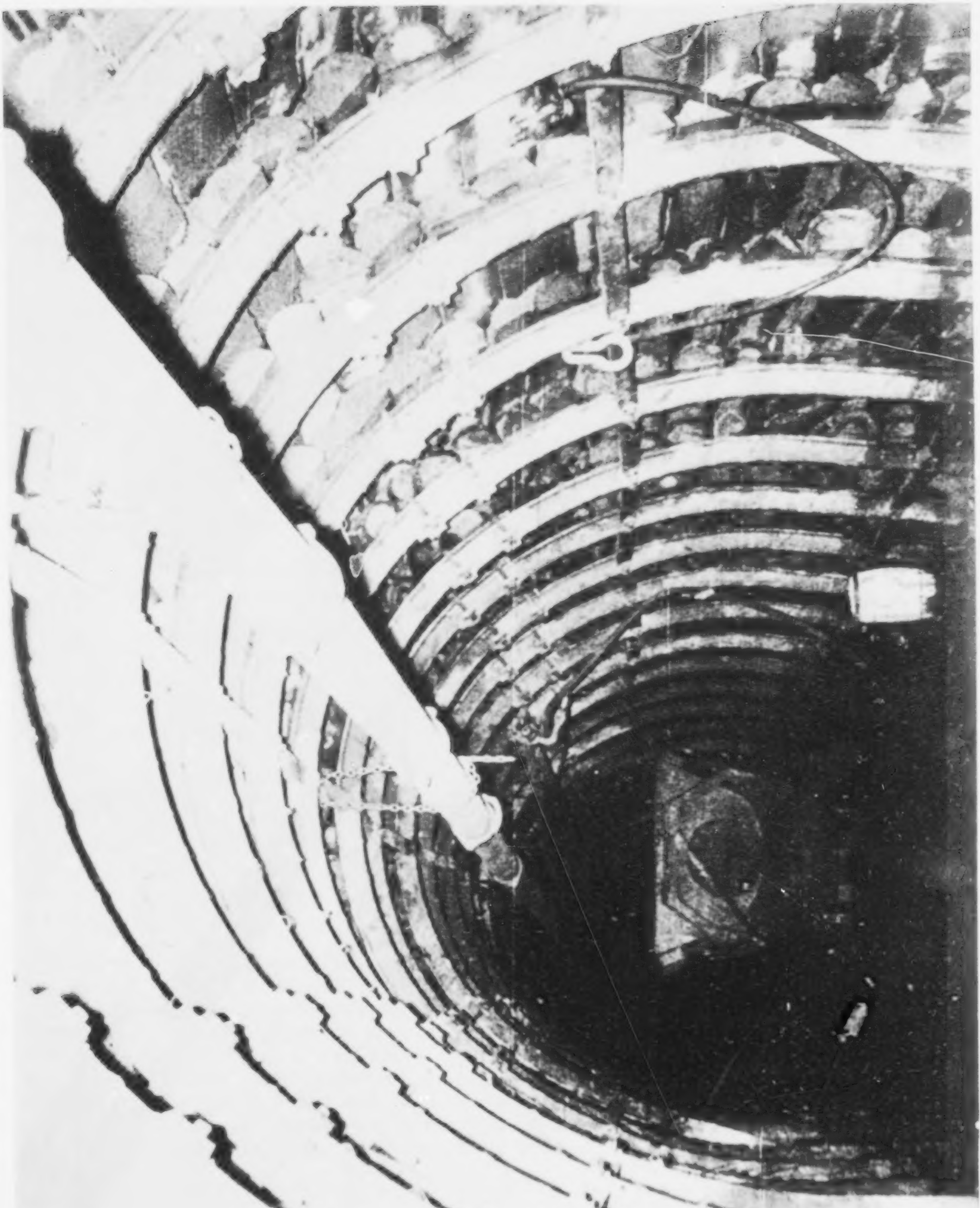

Figure 53. Heavy duty three-piece yielding arches

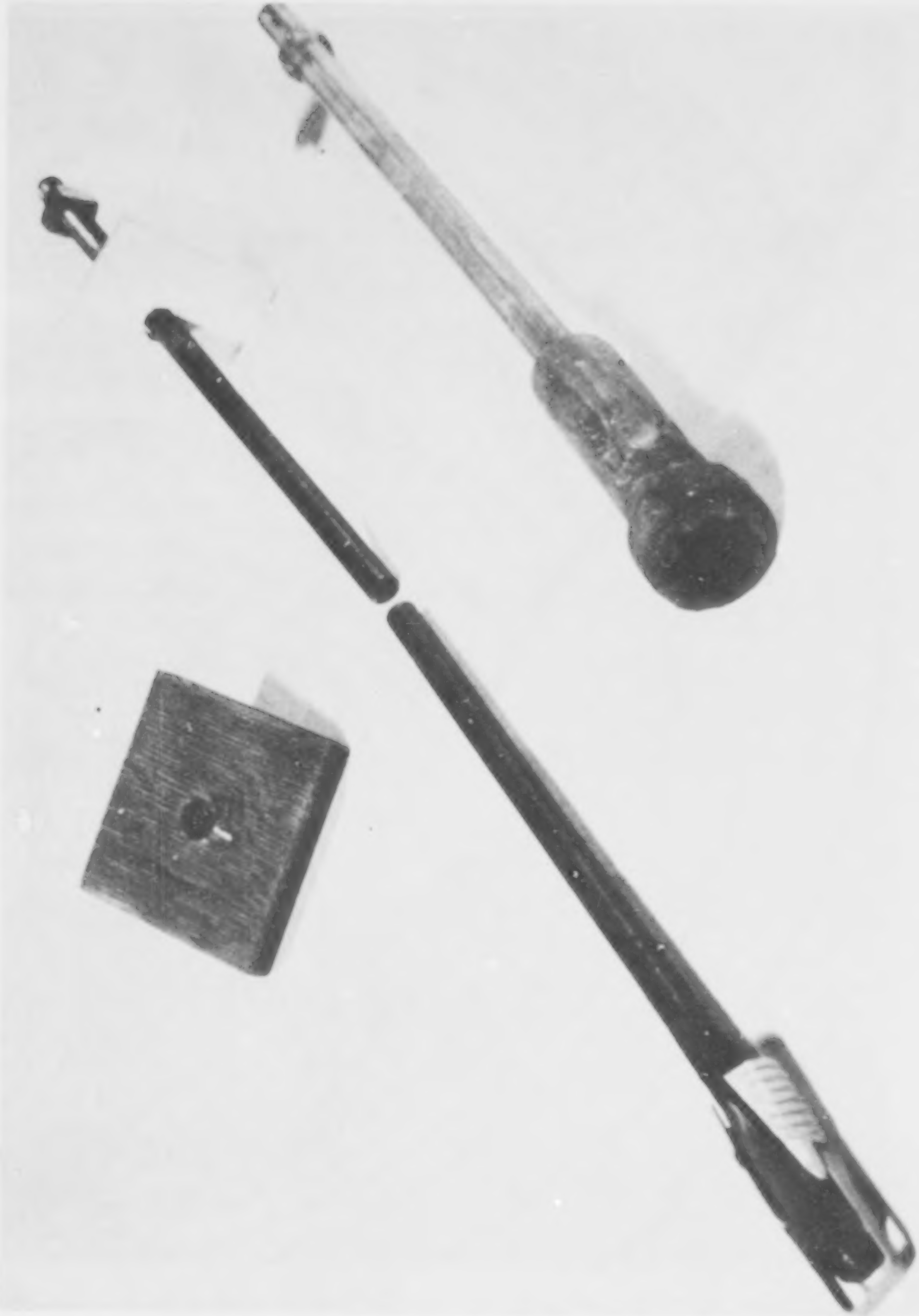

Figure 54. Rock bolts, complete

The rod, in 5/8" D for the 5' length and 3/4" D for the longer lengths, is made from 100,000 psi minimum tensile strength high strength steel, threaded for a length of 5 1/2" each end with a national cold-rolled left-hand thread. This left-hand thread is specified so that the bolts can be set and torqued with a rock-drill (stoper or jackleg). Each shipment of bolts is accompanied by a test certificate showing the breaking load of the threaded end of finished bolts thereby maintaining quality control by the supplier.

The expansion shell is the Stelco-Dempsey design and consists of two corregated forged steel leaves held together by a flat, spring-type cold-rolled steel bail which is spot welded to the leaves. The wedge-plug is designed to support the shell uniformly against the hole periphery and is made from Pearlite malleable iron which has great strength and abrasion resistance qualities.

The high-strength double-chambered square nuts are of standard size across the flats for both 5/8" D and 3/4" D bolts thereby simplifying the tightening procedure.

Rock-bolt plate washers are flat 4" x 4" x 3/8" mild steel and are supplied in convenient carry packs of 15 washers each. We have little if any recourse to use other accessories such as spherical seat washers, round dished washers, domed rock-bolt plates or hardened round washers.

Application and procedure

Throughout all our mines there is an extensive use of rock-bolts for mine support.

In development openings it is normal practice to install a row of 5-ft bolts in the back or roof of the opening for every 4-ft advance. The number of bolts per row varies with the width of the opening - usually 4 bolts for an 8-ft wide heading. If necessary, bolts are also installed in the walls to support slabby ground. All the bolts are torqued by rock-drills.

The length of bolts installed in stopes is left to the discretion of the mine supervisors. Generally the longer bolts, 6' x 3/4" D and 7' 6" x 3/4" D are used in the stope backs where heavily fractured ground requires stronger support and deeper anchorage. The standard bolting pattern is set at 4-ft centres but this spacing can be reduced if circumstances warrant a tighter pattern. Where necessary, the exposed pillar walls or foot and hanging walls are also bolted.

As soon as possible after exposing the new rock surface the area is scaled and rock-bolts installed. Holes are drilled to the required length with a stoper (for back holes) or a jackleg (for wall holes) normal to the surface or at right angles to any major fracture. The plastic sleeve is removed from the anchor, the leaves expanded and the bolt is inserted into the 1 1/4" D hole. The square nut is removed and a plate washer placed over the rod and the nut screwed onto the outer end of the bolt for about 2". A rock-bolt dolly is placed into the chuck of the drill and the driver head over the nut end of the rock-bolt which is then pushed into the drill-hole

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by the pneumatic leg of the rock-drill until the plate washer bears against the rock surface. The rock-drill is turned on and through its rotation of the dolly the bolt is tightened until the machine stalls. This method of tightening bolts imparts a torque of about 200 ft/lb to the bolt. Periodic checks with a torque wrench on newly installed bolts are carried out to ensure bolts are correctly installed. Using the above procedure one man installs 20 bolts, six ft long, in six hours.

Special rock-bolts

In ore and waste passes and U/G storage bins where attrition of the exposed nut end of the bolt would negate its usefulness grouted untensioned reinforcing bars have been effectively used. These bars, both 7/8" D and 3/4" D, have been grouted in such openings using polyester resins and sand-cement mixtures. Excellent bonding between the bar-grout-hole wall has been achieved and test pulls invariably result in failure of the bar and not the grout bond. In highly corrosive areas where the life of a conventional tension rock-bolt is limited the use of grouted bolts, by nature of its total encapsulation by the grout, has proven advantageous. The present major drawback for the wider application of this type of bolting is the cost and time factor. Although the bolt material is cheaper the polyester resin is expensive. Sand-cement is relatively inexpensive but the equipment to place it and the rapid set additive required is not. More important is the time element in that the ground should be secured without delay. Grout setting time can also delay the optimum work cycle.

Cleaned, tensioned and cement grouted reclaimed hoisting cable has been used successfully in lengths up to 120 ft to reinforce broken ground adjacent to an ore pass system (figure 55). In another instance the same technique was used to support the back of a highly fissured cut-and-fill stope.

Williams groutable tensioned bolts have been incorporated in the design of large permanent openings in depth, for example, a hoist room 76' x 34' x 36H at 4,025 ft below surface.

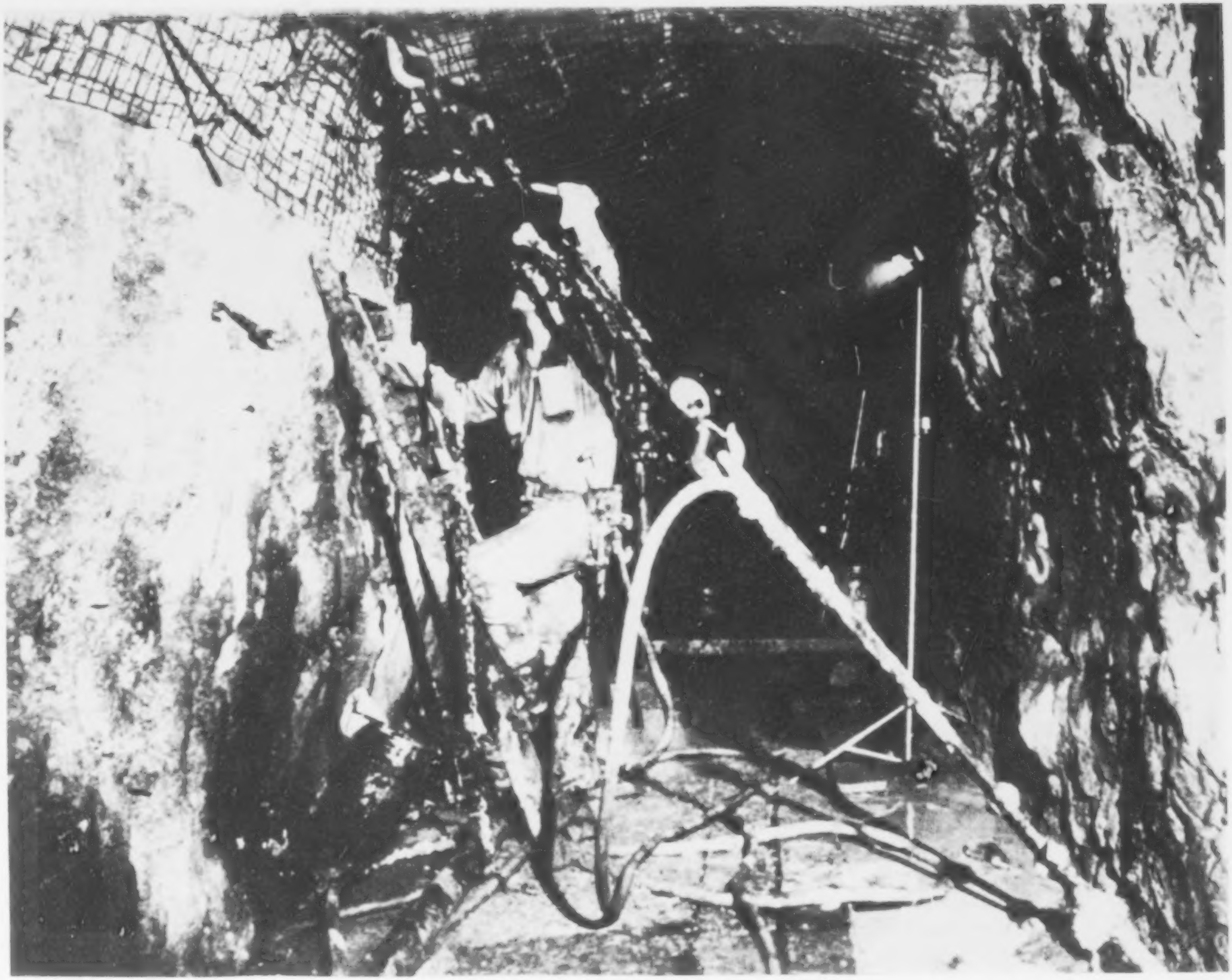
Cost

Rock-bolting is a labour-intensive method of ground support that is difficult to mechanize, particularly in our operations where 76 per cent of our production originates from cut-and-fill stoping. To rock-bolt the stope back from a broken uneven ore-pile with restricted headroom is no easy task, and to date no suitable piece of mechanized equipment has been devised to perform the job.

For the installation of a 6-ft rock-bolt, 62 per cent of the cost is for labour and the remaining 38 per cent for material in the form of the rock-bolt, plate washer, bit, steel and rock-drill maintenance charges.

In stoping where on an over-all average of one bolt is installed for 8 tons of broken ore, the cost of bolting is $0.69/ton. In our most common size development headings, 8' x 9', eight bolts are installed in two rows of four bolts each for eight feet advance, or one bolt/ft advance at a cost of $5.49/ft.

Figure 55. Tensioned cable bolts



5. Mesh

Electro-welded wire mesh and to a lesser extent expanded metal mesh are used extensively in our U/G working places. The purpose of this is to catch small loose material that frequently originates between adjacent rock-bolts, thereby making the work place safer and reducing the time spent on scaling (figure 56). These advantages by far offset the cost and become increasingly important in cut-and-fill stoping where the stope back may be 25 ft above the mucking floor prior to filling, and effective scaling without mobile scaling platforms is impossible.

Material and application

Weldmesh in 4" x 4" x 9 gauge and 3" x 3" x 9 gauge is supplied in 50-ft rolls, 5 ft in width, and used principally in stoping. In development work heavier weldmesh, 4" x 4" x 6 gauge, or expanded metal mesh, is supplied in 8' x 4' sheets.

Mesh is installed as an integral part of the rock-bolting operation, as the bolts are utilized to hold the mesh in place. The mesh is unrolled transversely in the stope and held up to the back by light mechanical or pneumatic props. The rock-bolt hole is drilled through the mesh and the bolt inserted with a 5" x 5" x 1/2" wooden washer at the outer end backed by the standard mild steel rock-bolt plate, the purpose of the wooden washer being to prevent cutting of the mesh wires by the rock-bolt plate when the bolt is tightened. With a 4' x 4' bolting pattern, there is a 6" overlap between successive courses of mesh. Where necessary, the mesh is extended down the pillar walls. After the broken ore is removed and the fill raised, the upper holes are drilled through the mesh, which provides a safe headcover during this operation. No attempt is made to remove this mesh before blasting the next cut. Tangled mesh in the broken muck-pile presents little problem where mucking is by Scooptrams, but this mesh must be removed at the crusher stations to prevent damage to belt conveyors. The problem of handling this scrap material together with rock-bolts and removing same from the mine has been overcome by installing locally designed bailing and strapping equipment for the mesh and hydraulic rock-bolt cutters at each crushing station.

Consumption and cost

The consumption and material cost at all mines in 1972 was as follows:

Item	Consumption	Unit cost	Total
3"x3"x9' gauge weldmesh	100,362 lin. ft	$0.426/lin. ft	$ 42,754
4"x4"x9' gauge weldmesh	705,429 lin. ft	$0.276/lin. ft	$194,698
4"x4"x6' gauge weldmesh	3,854 pieces (8x4')	$2.64/piece	$ 10,175
Expanded mesh	1,408 pieces	$6.56/piece	$ 9,236
Wooden washers	222,950 each	$0.07 each	$ 15,607
			$272,470

Material cost for 1 sq ft of stope-back, using rolled mesh, is $0.06, with labour costs for installation approximately $0.03/sq ft. Since in round figures 1 sq ft of stope-back generates 1 ton of ore, the cost per ton of the installed mesh is $0.09. This cost is well justified by the improved safety that the system provides.

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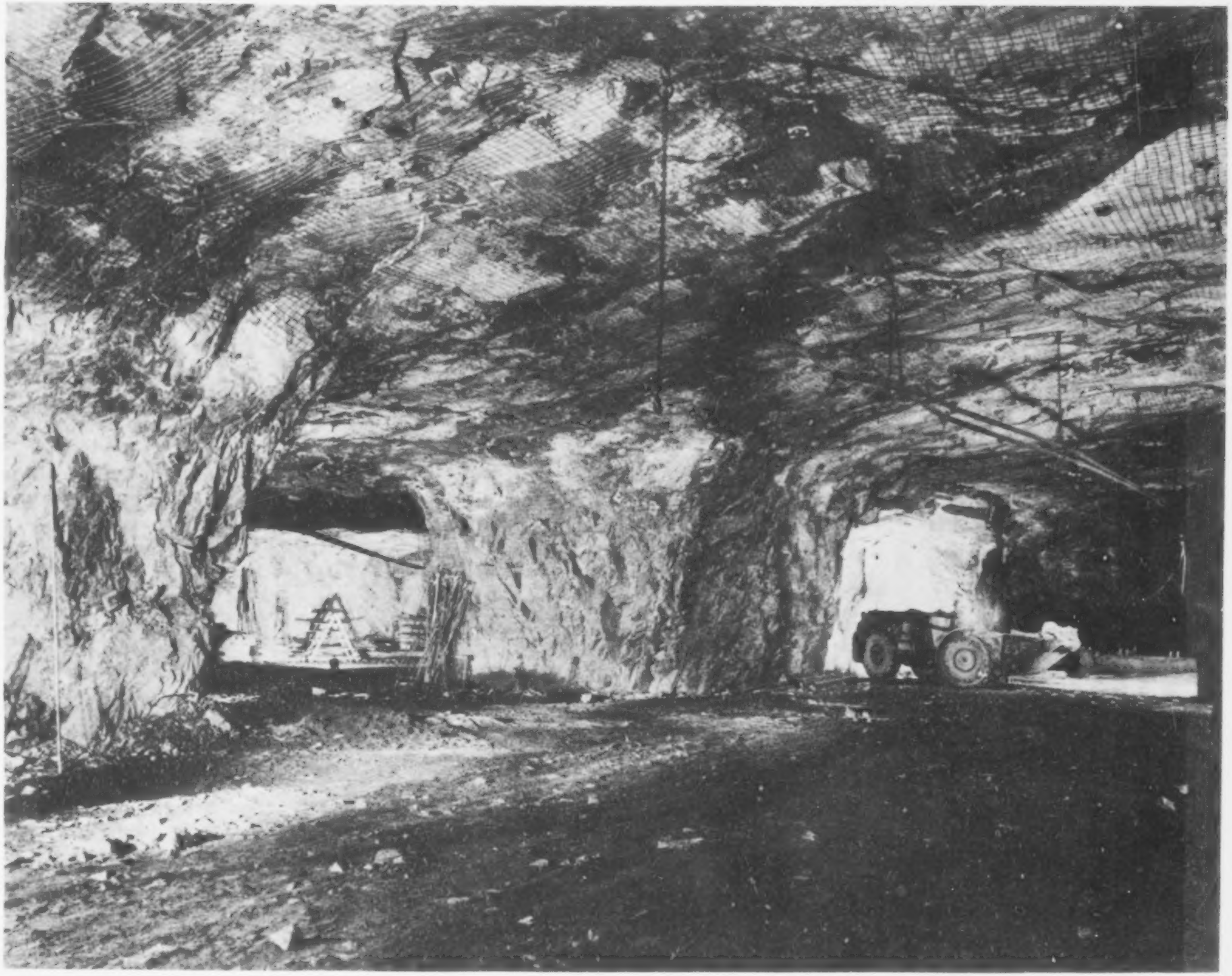
Figure 56. Application of Weldmesh



6. Gunite and shotcrete

The pneumatic spraying of aggregate, sand-cement mixtures to rock surfaces as a method of ground support has been practised at our mines for many years. This technique is finding wider application as our mines get deeper and the need arises for close support behind development faces in highly stressed ground. Other factors favouring this method of support are recent improvements in equipment design, the introduction of rapid set additives, increasing emphasis on safety, and the rising cost of materials for other forms of support, such as timber and steel.

In the following, gunite is defined as a mixture of -1/4" aggregate, sand and cement mixed in a cylindrical vessel with the required amount of water and blown through the delivery pipe in a wet state. No rapid set additive is contained in the mix. Shotcrete consists of -1/2" aggregate, sand and cement, to which is added a rapid set additive, and this mixture is blown dry through the delivery pipe with the necessary amount of water added at the discharge nozzle.

Material and equipment

Gunite. The mix consists of one part of Portland No. 1 cement to four parts of sand with a maximum particle size of 1/4" supplied in 50-lb capacity sealed plastic bags, stacked and strapped on pallets, thereby facilitating bulk handling. In this form with the mixture perfectly dry, it is possible to store the mixture on surface under all climatic conditions without any deterioration. The 50-lb bags are easily handled, split and dumped into the mixing chamber of the wet mix Model H True-Gun All machine by one man.

The Model H True-Gun All machines, of which we have three, are rail mounted, of the single-chamber design, and powered by an air motor. The mix capacity of the chamber is 9 cu ft and a rated production of 1 1/2 to 3 cu yds per hr.

Shotcrete. A typical shotcrete mix consists of 2,300 lb of 1/2" screenings to 800 lb of concrete sand which yields 1 cu yd to which is mixed 660 lb of Portland No. 1 cement. The aggregate must be clear of organic dirt and possess a uniform grading of particle sizes. The best results have been obtained with a mix containing 35 per cent to 40 per cent plus 4 mesh and not more than 5 per cent minus 100 mesh (Tyler). Also the aggregate-sand should have a moisture content of between 3 per cent and 5 per cent. A lower moisture content causes excessive dusting and higher moisture causes build up in the machine. The mix is delivered to the mine shaft by ready-mix trucks and transferred to mine cars for transportation to the shotcreting site. If the distance from the batching plant to the mine involves a travel time of one hour or more, the cement should be added at the mine prior to transferring the mix to the cars. Poor quality shotcrete will result if the cement is in contact with the moist aggregate-sand for a period in excess of 1 1/2 hours.

Two rapid-set additives have been used, Monoset T1 and Cyklosal B with rates of addition as follows:

15 lb per cu yd of premix for walls
20 lb per cu yd of premix for backs
25 lb per cu yd of premix for wet areas

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The additive is fed at controlled rates from a separate feed hopper, and care must be taken to prevent skin burns from this caustic material.

The Aliva 600 and the True-Gun-All Cyclone 61 are two shotcreting machines in current use. Both are mounted on rubber tires, air powered, and operate on the same principle. A circular multichambered drum rotates on a vertical axle and as the chambers pass beneath the mix hopper they are filled and, on rotating further, their contents are blown out into the discharge hose. The necessary water is injected to the emerging aggregate at the nozzle. The Aliva 600 has passed 4.5 cu yds/hr and the Cyclone, 61 2.5 cu yds/hr.

Application and performance

Gunite. This material is extensively used in drifts, x-cuts and raises to cover the exposed rock surfaces thereby providing a safe and maintenance free (no scaling) opening. Usually these openings are some distance away from stoping areas and not subjected to major ground movements which could cause cracking and subsequent deterioration of this relatively thin coating which averages about 3/8" thick.

A three man crew - nozzle man, machine operator and spare - spray an average of 721 sq ft of rock surface per man shift, or 83 linear ft of 8' x 9' drift. Rebound losses are about 14 per cent. During 1972 1,361,000 sq ft averaging 3/8" thickness were placed in our underground working.

Samples taken for uniaxial compression testing show the gunite to develop a strength of 4,500 psi after 28 days.

Shotcrete. The use of shotcrete is confined to larger permanent mine openings such as hoist rooms, where coatings of 4" thickness are employed to provide sufficient structural strength to withstand moderately high ground pressures. Other successful applications have been shaft station brows and shaft walls during sinking operations, walls and back of a subsurface multifan excavation complex, U/G electrical substations, and the back of a caving ore pass.

By using the rapid set additive a coating of 2" thickness can be placed in one pass even on running wet rock surfaces. The rapid setting qualities have the added advantage of providing early structural support to the exposed rock surfaces. The advantages derived from the use of this additive justify the cost at $0.45/lb.

Depending upon the system required to supply material to the shotcrete machine the crew size varies between four and five men. Furthermore the supply system has a very marked effect on the efficiency and cost of this operation as the following cost table shows. An average performance for a four-man crew is 200 sq ft of rock surface 2" thick per man shift. Rebound losses average 25 per cent.

Test samples show a uniaxial compressive strength averaging 5,000 psi after 28 days.

With both guniting and shotcreting it has been found that a skilled nozzle man and good lighting are essential if a successful application is to be achieved.

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Costs

A comparison between the cost of guniting and shotcreting, based on current labour and material costs and allowing for rebound losses is shown in the following tabulation:

Guniting: at 3.4 cu yd/man shift through machine

Costs/sq ft covered at 1/2" thickness

	(dollars)
Labour (including NWLC, incentive shift aux. and shift premium)	0.088
Material	0.076
Maintenance	0.004
Total	0.168

Shotcreting: at different quantity throughputs

Costs/sq ft covered at 2" thickness

Cubic yd/man shift through machine	3.7	2.0	1.0
		(dollars)	
Labour	0.128	0.238	0.472
Material	0.310	0.310	0.310
Maintenance	0.038	0.038	0.038
Total	0.476	0.586	0.820

The influence of the material handling system on the placement costs is illustrated by the above table, and in no instance is the rated capacities of either gunite or shotcreting machines achieved. The importance of improving the material-handlin system cannot be over-emphasized if throughput is to be increased and operating labour costs reduced to any substantial extent for these proven methods of mine support.

C. Summary

Throughout our underground operations the need for mine support to safely and economically maintain man-made excavations is an integral, important and costly part of the whole mining procedure. The various support media are utilized daily whenever their characteristics can be used to advantage.

The following table summarizes the form, cost and application of the principal support media:

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Table 3. Summary of data on support media

Support Medium	Form	Cost	Application
Hydraulically placed fill	Tailing	$1.10/ton of fill in place	Direct back fill in cut-and-fill stoping Delayed back fill in shrinkage and some blast hole
	Cement-consolidated tailing	$2.5/ton in place depending on cement: tailing ratio	Mucking floors Direct back fill in most mechanized cut-and-fill stoping UC and F stoping Delayed back fill in some blast hole stopes Localized areas where higher strength fill is required
Timber	Square (Douglas fir)	$300/1,000 board ft	Shaft sets Gangway caps
	Stull posts (pine and spruce)	Pine $120/1,000 board ft Spruce $200/1,000 board ft	Gangway posts UC and F stoping
	Lagging (pine and spruce)	Pine $120/1,000 board ft Spruce $200/1,000 board ft	Fill fences Minor support and covering
	Framed and flatted (pine and spruce)	Pine $120/1,000 board ft Spruce $200/1,000 board ft	Square sets Manway crib
Steel	Arches	$60/linear ft of gangway (installed)	Gangways
Rock-bolts	Expansion anchor 5/8"D and 3/4"D high tensile steel	$5.49 per ft (installed) of drift advance	Development openings
		$0.69/ton (installed) broken	C and F stoping
Mesh	Weldmesh	$0.09/ton (installed)	C and F stoping
Gunite and shotcrete	Gunite	$0.17/sq.ft. 1/2" thick (installed)	Development openings
	Shotcrete	$0.50-$0.80/sq.ft. 2" thick (installed)	Major long-term excavations

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Modifications to existing practical applications of support are continually being made and new materials and techniques evaluated.

By combining the expertise of production and technical personnel the challenge that nature presents for the future extraction of her mineral wealth will be met with safety and economy.

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VI. HARD-ROCK MINING IN METALLIFEROUS MINES

Ingvar Janelid

The paper deals with techniques and equipment influencing choice and layout of mining methods, both present techniques and developments that can be expected within the near future. Details such as rock mechanics, mining machinery, and environmental problems will be dealt with in other papers and therefore will be touched upon only in connexion with description of methods.

Hard-rock mining means the excavation of ores and rock which are so solid that drilling and blasting or in certain cases undercutting and caving are normally required. The present estimated quantities of ore and rock excavated in the world are as shown below (in million m^3).

Ore mining	
Underground	620
Open pit	1 550
Rock excavation and construction work	
Underground	130
Open pit	1 450
Total	3 750

Thus, the total annual quantity of ore and rock excavated in the world corresponds to one cubic metre per person. Because of the increase in population and increasing excavation of ore and rock per capita, the total quantity will increase rapidly.

When planning for new mines today, the most important question is environment and especially ventilation of fumes and exhaust gases. Mechanization of underground mines requires a completely new approach if efficient and good working conditions are to be attained.

Rapid development is taking place in the substituting of hard manual labour by mechanical operations, where manual effort is limited to certain manipulations executed by the worker. Automatic and remote controls of the entire working process have already been introduced in cement works, concentrating plants,

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and others. There are fully automatic systems for ore and rock transportation in the mines. As the capital costs are high, there will be an increasing need to make use of the installations effectively. High labour costs and demand for good environment conditions make it necessary today to mechanize as much as possible.

In comparison with other industries, pre-planning and automation in the mining industry are often hampered because existing conditions in and around an ore deposit are difficult to estimate, and as a rule, variable.

A. Drilling

The practical application of the various drilling methods is shown in diagrams. The application ranges, according to these figures, are valid for drills and bits available at present. When tungsten carbide and other materials are improved, the application ranges will change.

The work on the problems to design better and more efficient rock drills and associated equipment is proceeding continuously. Handheld machines with pusher legs are used in small drifts and under special mining conditions. During the last few years mechanized drilling for drifting and tunnelling, as well as for mine production, has been developed very rapidly.

As to long-hole drilling in, for example, sublevel stoping and sublevel caving, there has been rapid development, and the drilling capacity is up to 400 drilled metres per manshift. Even if production drilling is only a small part of all operations in mining, there is every reason to work on further improvements. The high production capacity is obtained by efficient and handy drilling units where one man is able to operate two or more rock drills.

I wish to emphasize the great importance of drill-hole direction by a diagram, which is taken to be valid for sublevel benching and a burden of 1.1 m. If the drilling is correct without deviation, it is possible to make total use of holes of any length. If the deviation at the bottom of the hole is 3 per cent, it is theoretically possible to make use of a drill-hole of 10 m to about 60 per cent, but with a 30-m long hole to only 20 per cent, compared to a straight hole. Deviation will lead to coarse fragmentation, and if it is important, failure in breaking the rock.

In precision drilling of raises the hole deviation is 1-3 per cent. The conclusions are that we have to operate the equipment so that drilling becomes accurate, and that lack of drilling precision limits the economic length of drill-holes.

For long-hole drilling, more rigid drill steel is now being used, mostly 1 1/4" round or hexagonal steel which also corresponds better to the heavier drills.

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In long-hole drilling, it is important to control the length of the holes. For this reason a handy instrument has been developed and is now on the market (ABEM-Atlas Copco). In principle, this electronic hole-length meter is an acoustic echo-sounder. The accuracy is about 0.5 m for a drill-hole of 25 m length. The inclination of a drill-hole can occasionally be checked with an accuracy of 0.1° (Finnish Peltonen inclinometer).

Figure 57 shows schematically a development programme for drill rigs and also indicates which automatic components are available today.

The question is whether demands are not developing faster than our planning. We must give high priority to details that will make it possible to place the operator in a cabin. Such details are rod-and-bit handling and automatic stop-and-return. Next step in the development will no doubt be remote control of the whole drill rig. This will require complete mechanization, automation of certain details, and also TV-supervision.

Drifting and raising

In small drifts, less than about 15 m^2 (160 sq ft), parallel hole blasting is mostly used. Long, large holes, drilled with down-the-hole drills have been discussed, partly as opening, partly for ventilation in, for example, sublevel drifts. In larger drifts, V-cut blasting is the most economical method. In many cases smooth-blasting of the profile is stipulated, especially if the rock is fissured and of a bad quality.

Raising has been subjected to an intensive development during many years with various types of drilling platforms, long hole raising with parallel cuts and so-called crater blasting as well as full face boring. I suppose that drifting with open raises as well as with Alimak raise climbers and Jora-lifts are well-known methods. A new diesel-powered Alimak raise climber has recently completed a 980 m long raise in Norway.

Long-hole drilling of raises has been developed during the last years and can still be realized thanks to improved drill rigs and the possibility of drilling straight holes to a length of over 50 m (164 ft). To succeed with raises of this length, the drill hole deviations must be kept under about 0.5 per cent, which means 25 cm (10") at 50 m (164') length. With the first method, blasting is carried out according to the technique for parallel hole blasting in drifting. In the crater method, fewer drill-holes, but with larger diameter, are used, and blasting is effected against the free rock surface in the raise. The advantage with this method is that there are fewer drill holes with larger diameters, giving a higher drilling precision. The crater method will probably make it possible to drill longer raises because drilling accuracy is better, and also accuracy is less important, as the explosive acts against the free surface and not against the drill holes.

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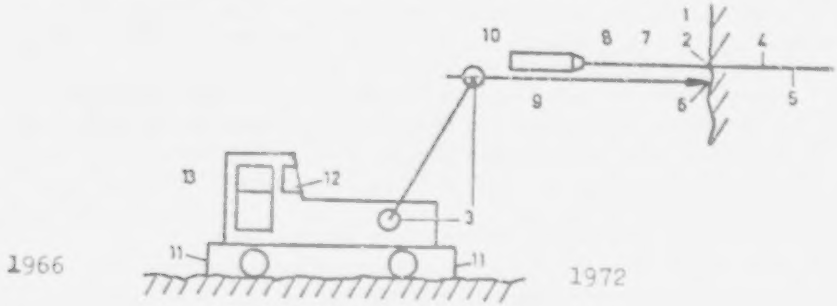
DEVELOPMENT PROGRAM FOR DRILLING UNITS



DEVELOPMENT PROGRAM	AVAILABLE	PRIORITY
1. Drill pattern	Projector etc.	
2. Collaring	-	4
3. Hole alignment	Parallel motion of booms	3
4. Hole deviation	Guide rod etc.	
5. Hole length	Hole-length meter	
6. Drill rod support	Different types	
7. Mechanized bit change	-	
8. " rod extension	For bench drills	
9. Feed control	" " "	1
10. Automatic return	Optional for various rigs	5
11. Stable rig	Standard	
12. Control panel	Certain rigs	
13. Operator´s cabin	" "	
14. Automatic stop	-	2

Figure 57. Development programme for drill rigs

Rock drills

In horizontal drifting, the well-known jackleg drills are still used to a great extent. Mechanized drilling has been successively introduced and more and more sophisticated rigs with hydraulic booms have been built. An example is the Atlas Copco Boomer S121 with a sound-proof cabin. The cabin makes it possible to work without ear-muffs.

An interesting development of drill rigs is the CAVO-DRILL 555. This rig is equipped with an extendable hydraulic boom in three variants so that the feed length can be changed between 1.0-1.6 m. Thus the same rig can be used for drifting, roof bolting and production drilling.

Hand-held drills and jacklegs are still used for production drilling, especially in small, irregular orebodies. Where mechanized drilling is possible, long holes are used, and today there are very efficient drills, mounted on more or less mechanized and automated rigs. In sublevel stoping where a large quantity of drill holes is produced for each set-up, column mounting is still used. In sublevel caving the drill rigs have rapidly been developed into mobile units for parallel hole drilling, for example, the Atlas Copco Simba 323. This rig has three drills which are operated by one man. The capacity is about 400 drilled meters per shift.

Drifting with large-diameter holes has many advantages: fewer small-diameter blast holes, longer advance per round, longer percentage advance and faster driving.

Full-face boring of horizontal drifts has not yet become economical in hard-rock mining. The method will certainly become more attractive when demands increase for environmental control and when bits can be produced which also stand up in hard rock. Other factors which can speed up the introduction of this method are speed of advance, 3-shift operation and adverse rock conditions at greater depth.

Most manufacturers of full face boring equipment use rotary heads with roller or disc cutters. Atlas Copco's Fullfacer uses another principle, called under-cutting, by which most of the rock is broken loose.

Blasting

As distinct from open-pit mining, factory-made cartridges of plastic or powdery explosives still dominate in underground mining. The reasons why the simple AN- explosives have not come into more general use are the problems with initiation and detonation stability in small-diameter holes, wet holes, and the fact that holes may be charged a considerable time before blasting. The development of new AN qualities, additives, better charging equipment, initiation equipment, and other improvements will certainly bring about an increased use of

It must be remembered that in hard rock generally the cost of a drill-hole is higher than the cost of the explosives charged into the same hole. From the economical point of view it is therefore very important to utilize the drill-hole volume as much as possible. Today there are available pneumatic charging machines with a high charging capacity, for cartridges as well as powder. Large truck-mounted pressure vessel charges of up to 750 L volume are used for AN/FO charging in the mines. There is also an ejector charging machine for direct mounting on a transportation container made of plastic. With a 3/4" charging hose it is possible to charge about 7 kg/min (15.5 lb/min) and to get a charge density of 0.95 kg/cm^3.

For charging with dynamite cartridges, there are pneumatic charging machines designed by Nitro Nobel AB. With the charging robot, a capacity of up to 1 ton dynamite per shift in long, small-diameter holes can be realized.

When charging drill-holes of average length, long cartridges of about 0.5-1 m (1-3 ft) are used to a large extent. For blasting of profiles or so-called smooth blasting, long cartridges of the explosive "gurit" is used. This explosive has a lower detonation velocity than dynamite and good detonation stability, even in small diameters.

Regarding ignition, great attention is paid to safety. An important development is the electric blasting caps, which require high ignition impulse. These caps are especially suitable when radio is used, since it is practically impossible to initiate them with radio waves. The present quality of AN/FO, together with pneumatic charging, has made it possible to ignite charges with only one standard cap. Perimeter blasting is also carried out with AN/FO in combination with detonating fuse. There are also special control instruments for ignition, lightning indicators and special cord blasting caps.

A controlled fragmentation of rock is essential for mechanization and automation of mining operations. This can only be obtained by controlled drilling, controlled charging and ignition, or to some extent by very large overcharging.

In normal benching the spacing of blast holes has been 1 or 2 times the burden. Recent investigations have shown that better fragmentation can be achieved if the hole spacing is increased to 6 or 8 times the burden. This drilling pattern has given better fragmentation in for instance cut-and-fill mining (figure 58).

B. Loading and transport

The methods of loading and underground transport considerably influence the organization and planning of mining methods and also the capacity per shift. In many countries there has been a rapid and remarkable changeover from rail to trackless transport, whenever possible. For instance in sublevel caving in big

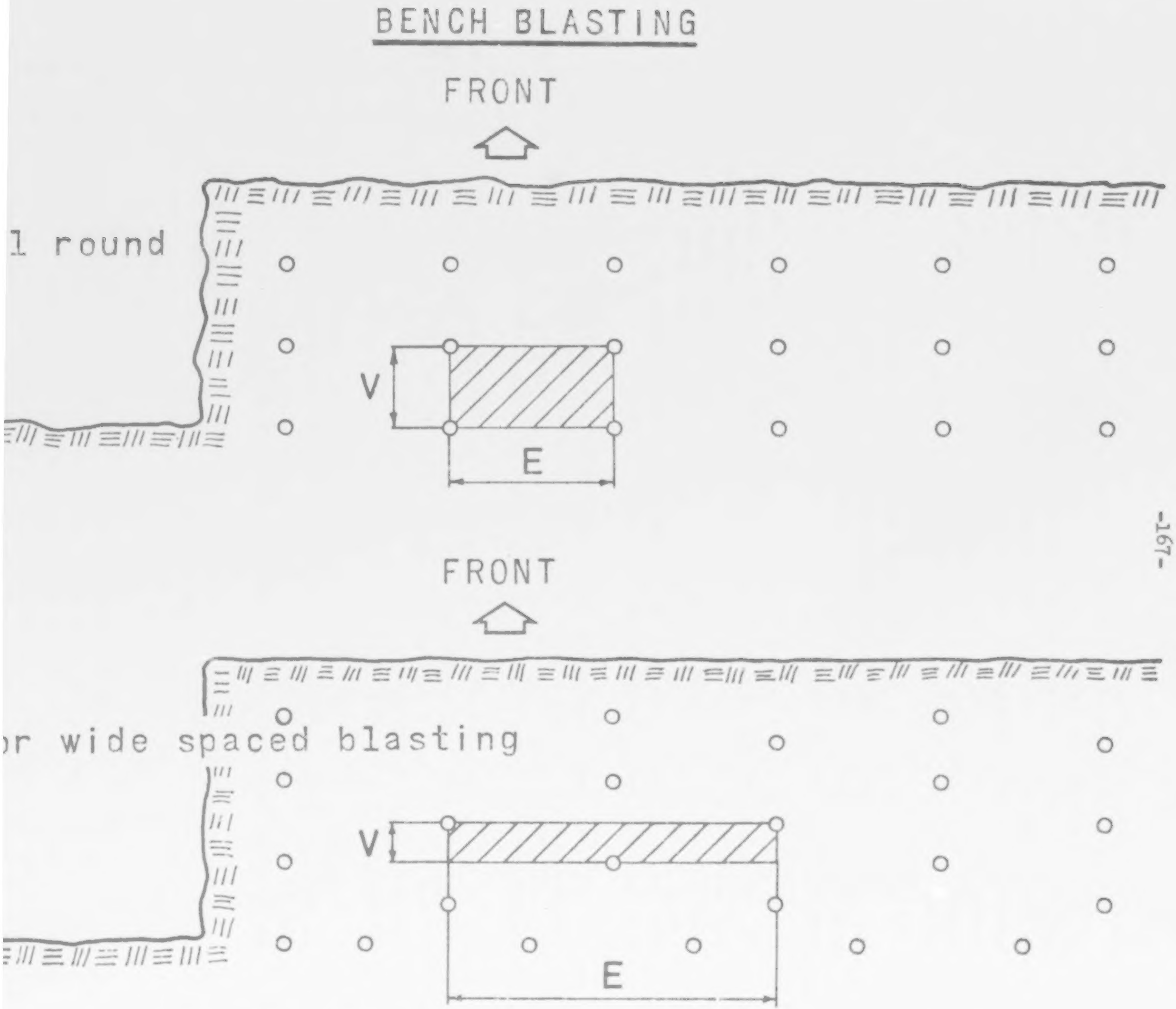

BENCH BLASTING
FRONT
1 round
V
E
FRONT
r wide spaced blasting
V
E

Figure 58. Drilling pattern, benching with wide spacing

Trackbound transport is still frequently used mainly on haulage levels for transport of ore from chutes to crusher or bins at hoisting shafts. In small mines with steep orebodies requiring proportionally long inclined drifts, trackbound transport is still adopted, but also in these cases there is a trend toward trackless transport. Machines and other equipment are moved between the levels through large raises in the orebody. The Hägglunds shuttle car is, however, an interesting alternative to trackless transport, especially in long drifts of small section, but also in certain mines.

The capacities of rocker shovel loaders have increased ten-fold during the last 20 years. Trackbound machines are still used in small drifts and for production loading, for example in sublevel stoping, block caving, and also in some sublevel caving mines.

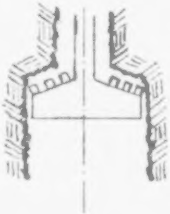
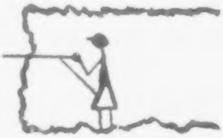
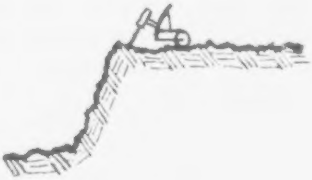
The compressed air-driven trackless rocker shovel loaders and LHD-machines are excellent for limited production, short hand distances and where adequate ventilation is difficult to arrange. However, diesel driven machines are superior todav, when conditions regarding size and dip of orebody, rock strength, etc. make it possible to drive large drifts (at least 3 x 3 m), when inclined drifts for moving equipment can be arranged and sufficient ventilation can be installed at reasonable costs.

If the above conditions can be met, there is no reason to hesitate at introducing big diesel driven units also in small orebodies, where the annual production is limited. However, some extra planning and a preventive maintenance system are required in order to keep up continuous production.

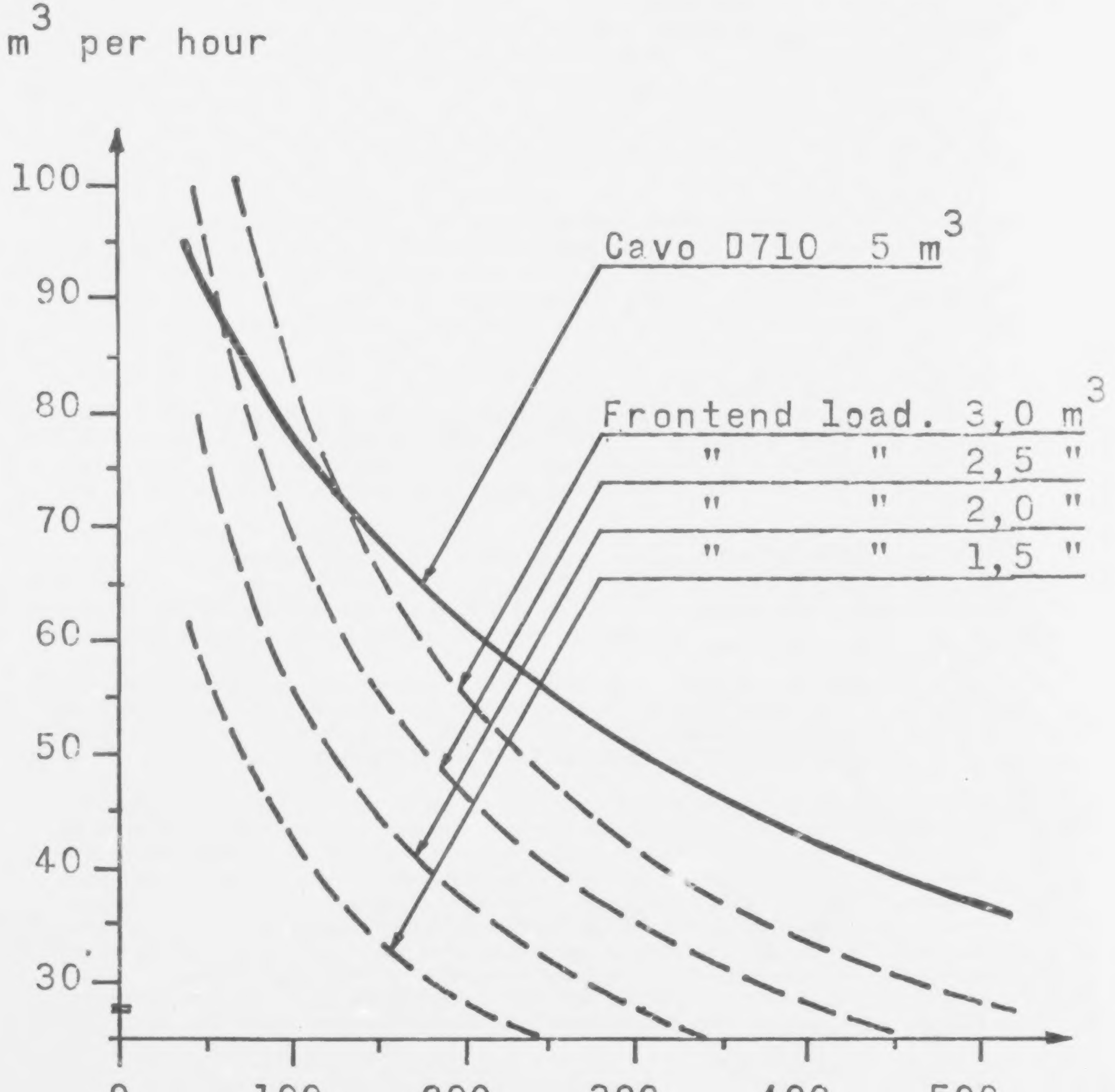
A new diesel-driven LHD-machine, CAVO D 710 (figure 59) is different from other loaders in many respects. For a given capacity, the required drift size is smaller. It has a hydrostatic drive which gives less peak load of the engine and therefore less exhaust volume. The hydrostatic drive greatly simplifies manoeuvring and is very suitable for remote control.

An interesting new machine is the Häggloader. Loading is effected by gathering arms which move vertically as well as horizontally. The rock is brought to a conveyor from the front and from the sides. The present loader can be used in drifts of 2.5-3.5 m width and a minimum height of about 2.5 m.

Transport with diesel trucks in inclined drifts has a special influence on mine planning. At Tuolluvaara, Sweden (figure 60), 3 Kiruna-trucks K 500 with a net load of 45 tons are presently used for hauling ore from levels below the crushing plant. The inclination of the haulage drift is 1:10 and the speed is 12 km/hr upgrade and 20 km/hr downgrade. This transportation technique will certainly be very important in the future, in planning of new mines as well as mining below existing crushing plants. Of course, there are certain disadvantages such as expensive haulage drifts and ventilation problems, but there are also many advantages. The incline can be driven continuously in step with the mining and ore can be hauled directly from different mining areas to crushing plant or skip bin.

METHOD	S.crowns/m^3
Diamond drilling	50.000
Hammer drilling	5.000
 Full face drilling	500
 Drifting	50
 Bench blasting	5
Coarse crushing	2,5

LOADING AND HAULING CAPACITY



Driverless transport from loader to orepass is now in successful use. An automatic steering and control system has been designed by Saab-Scania and mounted on a Fiskars dumper. This system has been tested by the Boliden company.

C. Rock mechanics

Rock mechanics is the science by means of which mining men hope to be able to solve various problems of mining as well as stability and safety. However, most of the problems are more complicated than usually expected.

The application of rock mechanics in mining can be grouped as follows:

Geological conditions	Type of rock and ore Tectonics
Fundamental problems	Strength, stability Fragmentation Gravity flow
Measurements	Theories Laboratory measurements Field investigations
Technical problems	Drifts, excavations Mining methods Caving
Results	Technical, economical solutions Safety

As mining goes deeper, the knowledge of rock mechanics will be more and more important for the solution of technical-economical matters. The problems of measurement and calculation are considerably more difficult in mines with their very inhomogeneous material than for concrete and steel construction.

Destructive as well as supporting processes exist in mining. By controlled blasting the destruction of the rock can be limited and by grouting, rock bolting and concrete guniting, weak parts can be strengthened. The details need not be gone into, as they are well-known methods and well described in the literature.

The measurements applied in mines are rock pressure measurements, strain measurements of various kinds as well as microseismic measurements. Rock pressure measurements with freeboring give direction and size of rock pressure at certain points. Deformation measurements with long wires, rod extensometers and levelling indicate movements within larger or smaller areas, while "microseis" measurements can be applied as warning signals, where collapsing is going on or might occur. At present an automatic equipment for "microseis" measurements exists and there are plans for a central control and registration of "microseisms" in a

One thing, however, we have to bear in mind: it is not possible with standard supporting means to resist high rock pressure when the natural rock support is taken away; it is a combination of artificial support and yield in walls and pillars which stabilizes the mining areas. At great depths with high rock pressure one solution of problems connected with this matter is to direct the pressure from the working area and face through stress relieving slots in the rock.

D. Mining methods

Underground mining methods are usually classified or distinguished from each other by the type of support required in the stope. Supports required may range from almost nothing to heavy timbers supported by sand or mine waste rock. Even in the strongest rock large underground openings will cave eventually. If the ore is weak, it caves readily. A system of stoping is sometimes used that takes advantage of this weakness. Mining methods usually fall into one or more of the following systems.

Underground mining methods

Mining system	Method
Open stoping	Room and pillar Sublevel open stope (Glory hole) (Gophering)
Filling	Horizontal cut-and-fill Inclined cut-and-fill Undercut-and-fill Square set (Timbered slot) (Resuing) ------- Shrinkage
Caving	Sublevel caving Block caving Induced block caving Longwall caving Top slicing

In the room and pillar method, 10-30 per cent of the mineral remains in the pillars. To recover the pillars, temporary support can be put in before the ground is allowed to cave. This procedure is called retreating mining because after the pillar recovery starts, no attempt is made to go back into the block.

The sublevel open stope system is mostly used in steep orebodies. The ground, both walls and ore, must be strong enough to stand unsupported over large spans. To start mining, it is first necessary to cut a slot across the ore, from footwall to hanging wall. This open slot provides a free face for blasting and the ore is thrown into the open stope. The ore can be removed from the stope in several ways, for example through chutes, by slushers or mechanical loaders.

By horizontal cut-and-fill mining a cut of ore is mined and waste is brought in to support the walls of the mined out portion of the stope. Detailed description will be given later. The cut-and-fill method can be used only if the ore is fairly firm and the walls will stand unsupported until waste fill is brought in. Sand is now commonly used instead of waste.

Inclined cut-and-fill was a rather common method before mechanical means were used in the stope to take the ore out and to bring the waste in. This system is seldom used today but can work efficiently in special situations in small mines.

Undercut-and-fill is a rather new method used for mining weak orebodies which do not allow horizontal cut-and-fill. In undercut-and-fill, drilling, blasting and loading are carried out underneath an artificial roof. This roof can be sandfill stabilized with cement or even occasionally concrete.

The square set method can be used in both narrow and wide orebodies and the fill material can be sand or mined out waste rock. The method is nowadays only used under special conditions, such as very weak ground, valuable ore, sufficient supply of timber.

The resuing method is occasionally used for high grade ore in very narrow veins when it is necessary to take out extra waste rock in order to get sufficient space for mining. The resuing method is an attempt to keep ore and waste separate during mining.

The shrinkage method can be employed in steeply dipping veins and orebodies if the ore and the walls are strong and self-supporting. Broken ore is left in the stope to provide a working platform for the miners. Because breaking the ore increases its volume by 40 per cent or more some ore must be drawn from the stope as mining progresses. The shrinkage method has been very widely used but is difficult to mechanize and of lesser importance today.

Sublevel caving has become one of the most important mining methods today. The method is mostly used in wide orebodies but also in steep dipping narrow orebodies. The advantages of sublevel caving are mainly that it consists of well standardized methods for drifting and long-hole drilling which facilitates mechanization and gives a safe operation since all work is carried out in drifts under controlled roof. There are also certain possibilities for selective mining which do not exist in, for example, block caving. The disadvantages of sublevel caving are that ore losses and dilution are relatively high.

The ore losses vary with the properties of ore and waste rock, shape of orebody etc. An economical aspect of importance is the possibility to separate the waste rock from the ore before secondary crushing and grinding.

The block caving method can be used for big orebodies and if the ore caves readily. After one block is developed the ore breaks by itself and has not to be drilled and blasted. Block caving is a large production low-cost method. If an opening is large enough it will eventually cave even in the firmest and strongest rock, but a caving system requires that the ore or rock will cave over a small unsupported area. The block caving method is widely used for mining big orebodies containing low grade ore and sometimes for high grade orebodies underneath low grade ore which is mined to a certain cut-off grade.

Induced block caving has been introduced at places where standard block caving methods result in too coarse fragmentation. Induced block caving will probably be used more in the future when the weaker parts of big orebodies have been mined out with block caving and where other methods such as sublevel caving are not suitable.

Longwall caving is a common method in coal mining next to the room and pillar method. Longwall caving like sublevel caving may also be used in metalliferous mines if the ground conditions are appropriate.

Top slicing is classified as a caving method because the overhead waste rock caves downwards. The ore itself does not cave but has to be drilled and blasted. The method is seldom used nowadays because a heavy support by timber is needed, and the production per manshift is rather low.

Room and pillar mining is rarely used in small irregular orebodies. On the other hand, room and pillar mining has been developed into an efficient and highly mechanized method for large horizontal deposits with competent rock.

As an example we can mention Laisvall, Sweden, where a so-called low front method is used. A roof tunnel is first driven, and then the lower part of the stope is excavated with benching. Parts of the remaining long pillars are recovered by cross-cutting.

The degree of mechanization and the size of equipment depend on the stope size, which in turn depends on the roof stability. By means of the Finite Element Method, deformations and stresses in roof and pillars may be calculated in order to get the best mining layout.

For room and pillar mining in orebodies dipping about 12-45°, Alimak-Verken have developed a new piece of equipment, Groundhog. It is based on the type of equipment used for raise climbers. The method is intended for orebodies whose dip is too steep for trackless equipment and where broken rock will not move by gravity down to a loading level.

Sublevel open stoping is used in many mines all over the world. In principle, this method is simple and only a few examples will be mentioned.

Mining at Strossa, Sweden (figure 61), is a modified open stoping. The ore between 135 and 195 m levels is divided into transverse stopes of 22 m width and 12 m wide pillars between the stopes. In the first step a 10 m thick roof is left over the stopes. Drilling for excavation of the stopes is carried out partly from the main haulage level 195 m, partly from an intermediate level, 170 m. Loading takes place at 195 m level directly into trackbound haulage cars.

In the second step the roof and the upper parts of the pillars are blasted down and also mucked out at 195 m level.

Lastly the remaining lower parts of the pillars are mined out by sublevel caving.

Drifting equipment consists of Atlas Copco Zax 202 for drilling, LM 250 and 4 m^3 Granby cars for loading and transport. Long-hole production drilling is done with Atlas Copco Simba 25, using BBC 120 drills and Sandvik Coromant 1 1/4" extension drill steel. For production loading, Atlas Copco LM 250 and 4 m^3 Granby cars are used. Each train consists of nine cars, and one man handles both loader and train. The trains are remote-controlled at loading and dumping stations.

Production underground is about 80 tons per man and 8 hour shift. The cost distribution is as follows (in percentages):

Overhead costs	26
Development and mining	58
Loading, haulage, crushing, hoisting	16
	100

Work-hours in 1971 had the following breakdown:

Mining	39
Electrical, mechanical, timbering etc.	24
Track-laying, pipe-laying, materials handling	15
Drifting, raising, shaft sinking	7
Track maintenance, cleaning	5
Loading in drifts	5
Haulage, crusher and skip supervision	5
	100

An interesting application of open stoping is found at Vintjärn. Figure 62 shows the principle of drilling. By using drilling upwards as well as downwards, even a very irregular orebody can be mined.

At Idkerberget, Sweden (figure 63), an orebody below 800 m depth is mined by open stoping, since orebody as well as country rock have good rock mechanical properties. Drilling is done partly from drifts in the very competent footwall but charging and blasting comprise only those parts of the holes which are located in ore.

INCLINED TRANSPORT
TUOLLUVAARA

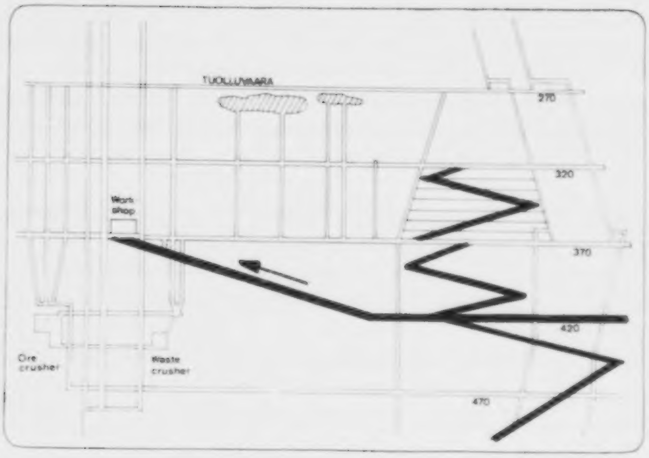

TUOLLUVAARA
270
320
Work shop
370
420
Ore crusher
Waste crusher
470

Figure 61

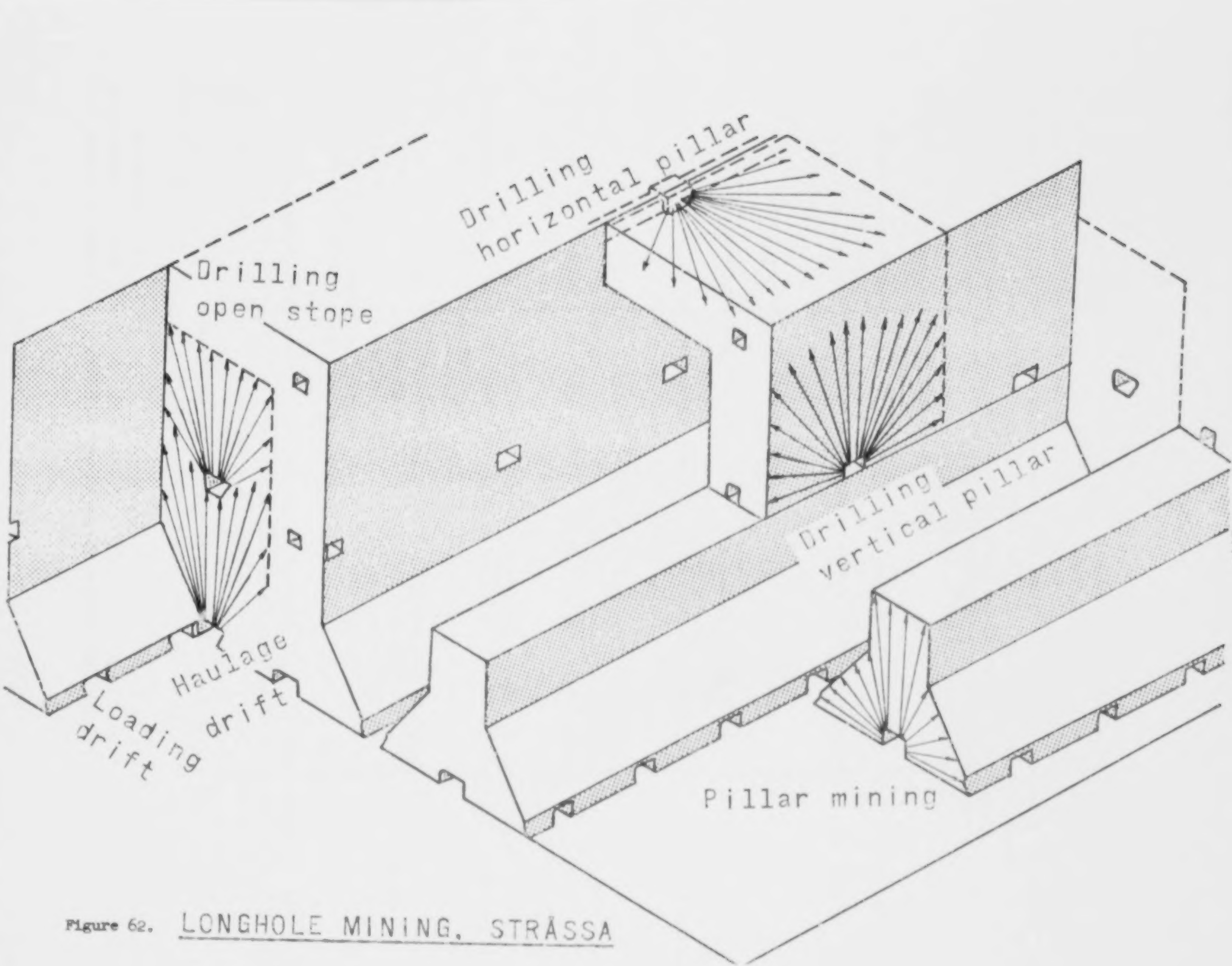

Drilling
horizontal pillar
Drilling
open stope
Drilling
vertical pillar
Haulage
drift
Loading
drift
Pillar mining

Figure 62. LONGHOLE MINING, STRÅSSA

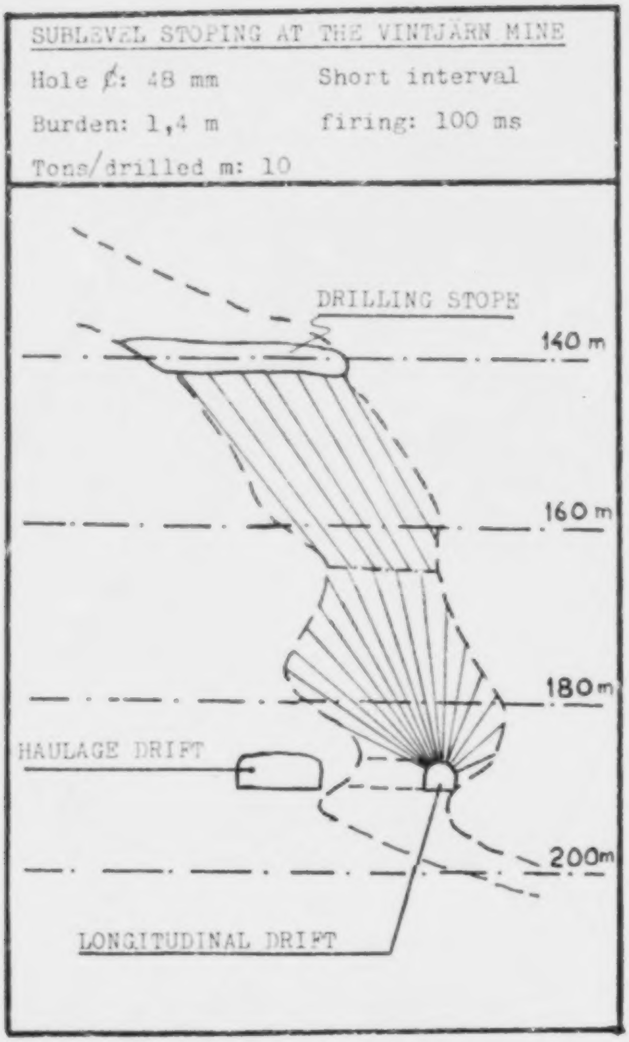

SUBLEVEL STOPING AT THE VINTJÄRN MINE
Hole Ø: 48 mm
Burden: 1,4 m
Tons/drilled m: 10
Short interval
firing: 100 ms
DRILLING STOPE
140 m
160 m
180 m
HAULAGE DRIFT
200m
LONGITUDINAL DRIFT

Figure 63. Sublevel stoping, Vintjärn

Cut-and-fill mining

In recent years the importance of cut-and-fill mining has increased and will probably continue to grow. The main reasons are the ever increasing environmental demands, which in many cases make it impossible to use caving methods, and also the attractive possibility of disposing part of the tailings underground instead of in large ponds on the surface. Another reason is that mining reaches greater depths, which means that cut-and-fill mining or at least backfilling can be a condition for managing the mining problems. Furthermore, cut-and-fill mining generally causes little dilution and almost complete recovery of available ore quantities which can be of great advantage in case of eventual mineral shortages.

Some general development tendencies of cut-and-fill mining are as follows:

(a) Change-over from dry waste-rock filling to hydraulic filling, mostly with tailings;

(b) Increased use of cement stabilization of hydraulic fill;

(c) Change-over from trackbound to trackless loading and transport;

(d) Increased use of ramps instead of raises and orepasses, for communication and transport.

As is well known, cut-and-fill mining means that the excavated stopes are backfilled with some kind of filling material. The different materials used today are shown in a table (see figure 64). The different functions of the fill are shown in figures 65-67.

The fill has a stabilizing effect on surrounding rock, and in order to investigate the effect of different fill materials, Finite Element calculations have been carried out.

The most common filling method today is hydraulic filling whereby sand, mixed with water, is pumped or flows by gravity through drill holes and pipes down to the stopes. The sand mostly used is tailings from the concentrator which are passed through cyclones, where the finest particles are separated and a permeable and stable fill can be obtained.

In certain cases, cement stabilization is used for two main reasons. Firstly, only the surface layer may be stabilized, that is, in the final stage of a filling, and cement is added to the sand-water mixture so that the last foot or so of the fill will be stabilized. This gives a hard and smooth surface, suitable for trackless equipment, which also separates broken ore and fill so that no ore is lost in the fill and no fill is mucked out together with the ore.

Secondly, the whole fill may be stabilized with cement, which gives a rather solid mass so that a wall of fill material can be exposed and stand up as support. This makes it possible to utilize special mining methods such as drifting and

/...

SUBLEVEL STOPING, IDKERBERGET

LEVEL:	810-870 m
DRILL HOLE DIAMETER:	48 mm
BURDEN:	1.5 m
TONS/DRILLED METER:	9.0

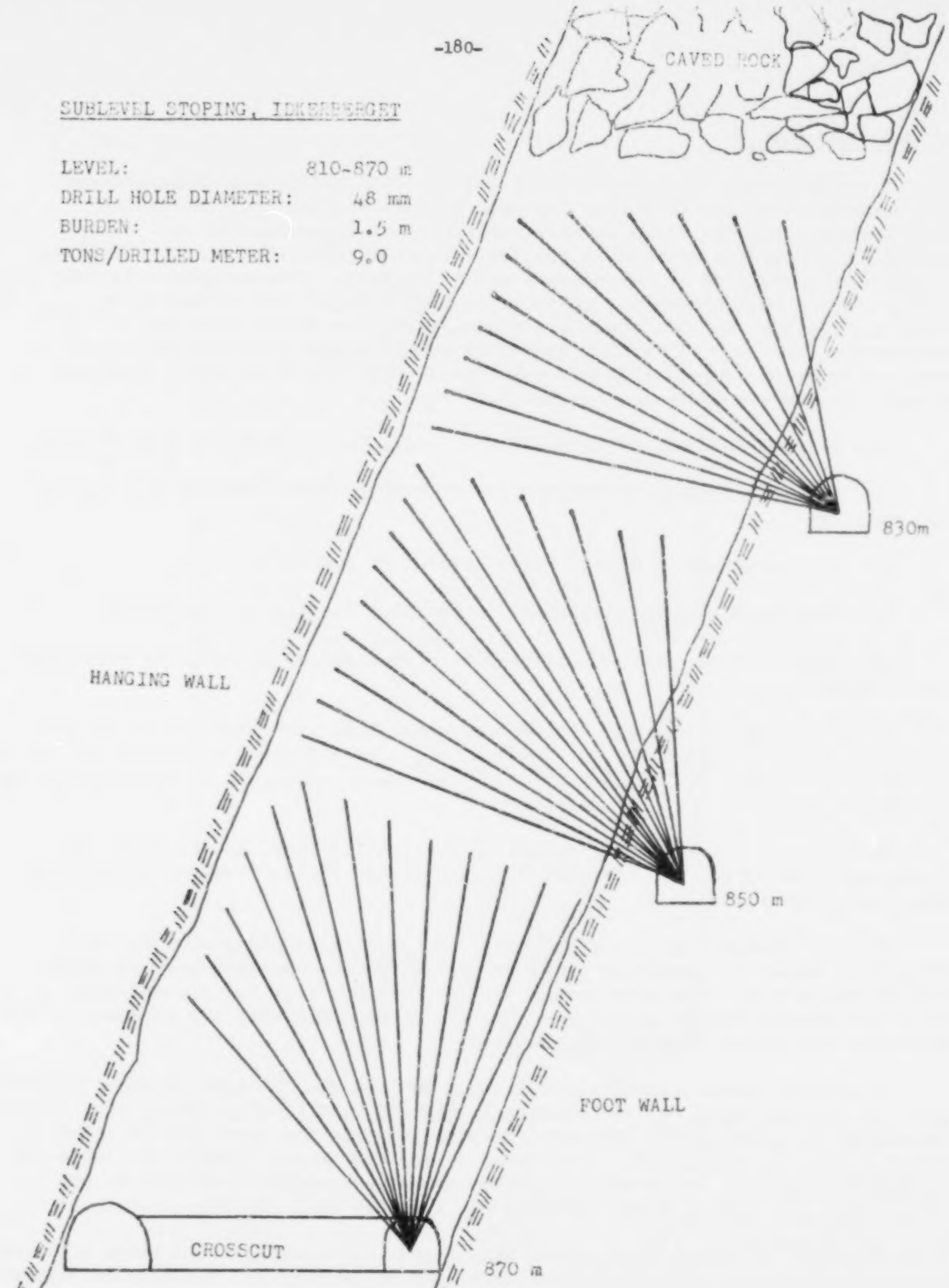

CAVED ROCK
830m
HANGING WALL
850 m
FOOT WALL
CROSSCUT
870 m

Figure 64

complete filling, undercut-and-fill and other types of pillar mining. A very important factor is the utilization of the finest particles of flotation tailings. This can partly be realized by means of special flocculents.

Mixing of sand, water and cement, if any, is done in mixing plants which can be more or less automated. Fully automatic mixing plants are used, for example at INCO and Falconbridge, Canada. Pulp density as well as cement contents are governed from a control panel.

Applications. Figure 65 shows schematically how different types of cut-and-fill mining can be applied, depending on the rock properties.

A suggested method of pillar mining at Falun Mine, Sweden, is shown in figure 66. The method is called drift-and-fill as distinguished from normal cut-and-fill. When a stope/drift has been mined out, it is filled up to the roof with preferably cement-stabilized hydraulic fill, and when the fill has drained and set, the adjacent part can be mined with a new drift. The previously filled drift supports the roof and safe working conditions are obtained. When the whole pillar area has been excavated with a number of drifts, mining continues on top of the filled drifts, and a new slice can be taken out.

Undercut-and-fill is shown in figure 67. The principle is to create an artificial roof. On the floor of the mined out stope some kind of reinforcement is placed and then the stope is filled with cement-stabilized hydraulic fill. At Kristineberg, Sweden, the reinforcement consists of half logs, placed transversely over the stope, a reinforcing net on top of the half logs and bolts through the half logs and into the fill or the hanging wall. When working under the artificial roof, the half-round logs can be supported by timber if necessary. This method is used in many mines in Canada, Australia and Africa.

Finally, an application should be mentioned which will probably increase in future, namely backfilling of stopes which have been mined by some other method, e.g. sublevel stoping or room and pillar. Here, backfilling is carried out in order to increase stability of the area, prevent or reduce cave-ins and subsidence on the surface, or simply in order to get rid of tailings underground instead of creating an environmental problem on the surface.

Sublevel caving

At present, sublevel caving is the most widely used mining method in Sweden. Drift dimensions, sublevel height, etc. vary considerably even within individual mining areas.

Figure 68 shows the development of sublevel caving from the first tests to what I hope will be normal within the next few years. From real sublevel caving with the blasting of short holes, which results in induced caving of the ore, the development has gone towards even longer drill holes. The drill patterns have previously been fan-shaped, partly because it was thought that

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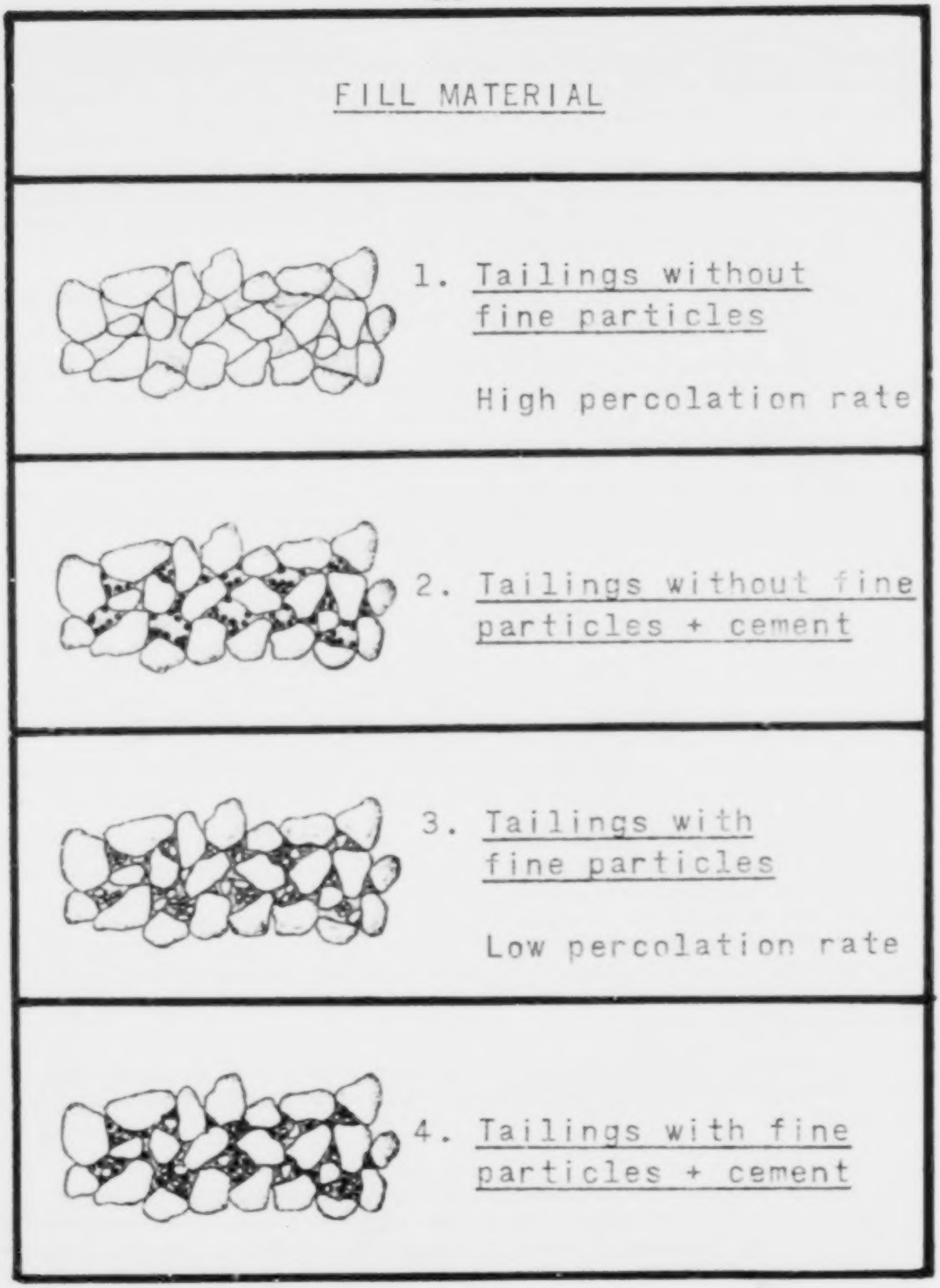
FILL MATERIAL
1. Tailings without fine particles
High percolation rate
2. Tailings without fine particles + cement
3. Tailings with fine particles
Low percolation rate
4. Tailings with fine particles + cement

Figure 65

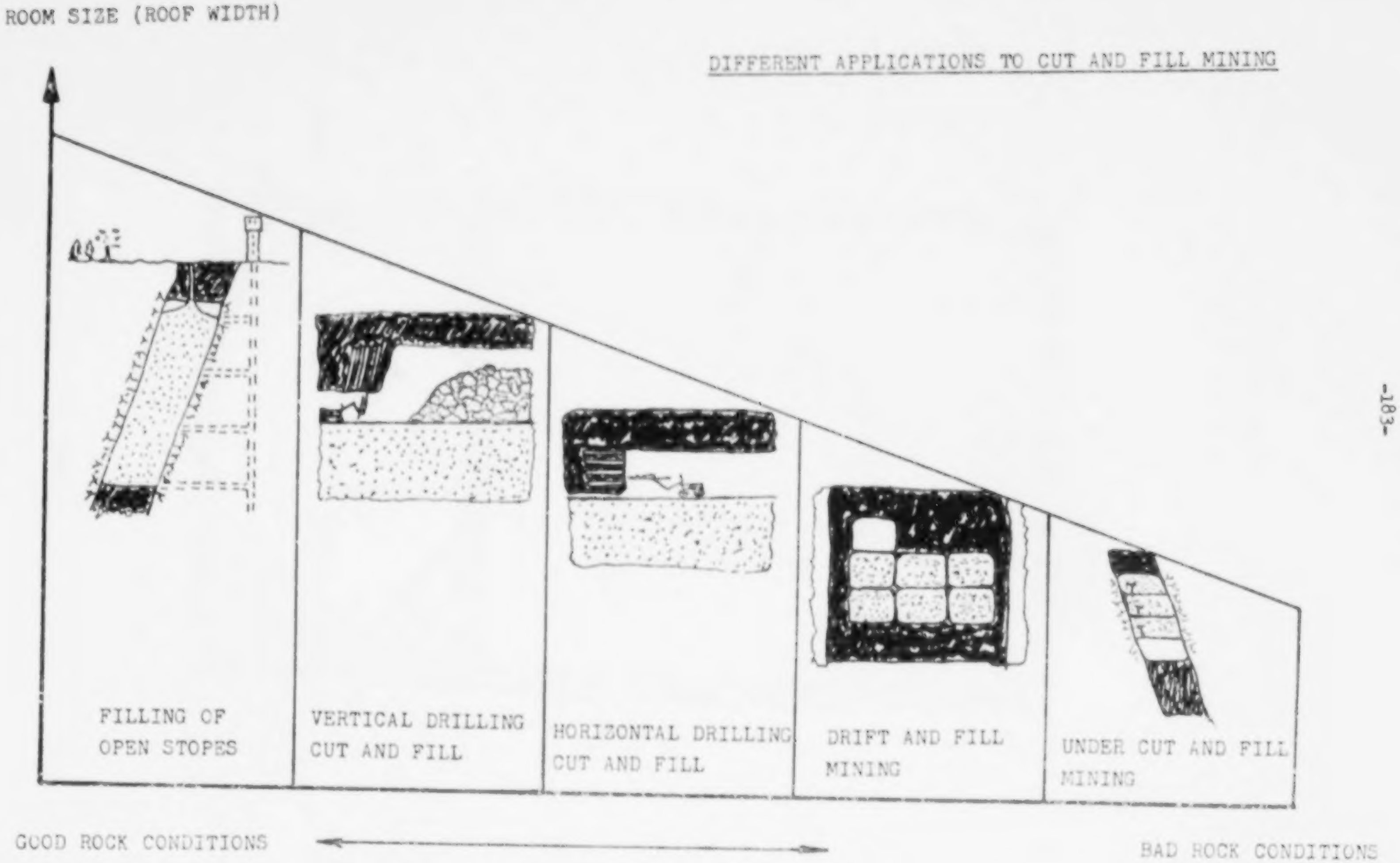
ROOM SIZE (ROOF WIDTH)
DIFFERENT APPLICATIONS TO CUT AND FILL MINING
FILLING OF OPEN STOPES
VERTICAL DRILLING CUT AND FILL
HORIZONTAL DRILLING CUT AND FILL
DRIFT AND FILL MINING
UNDER CUT AND FILL MINING
GOOD ROCK CONDITIONS
BAD ROCK CONDITIONS

Figure 66. The use of various filling methods

DRIFT AND FILL MINING

DRIFTING WITH COMPLETE FILLING

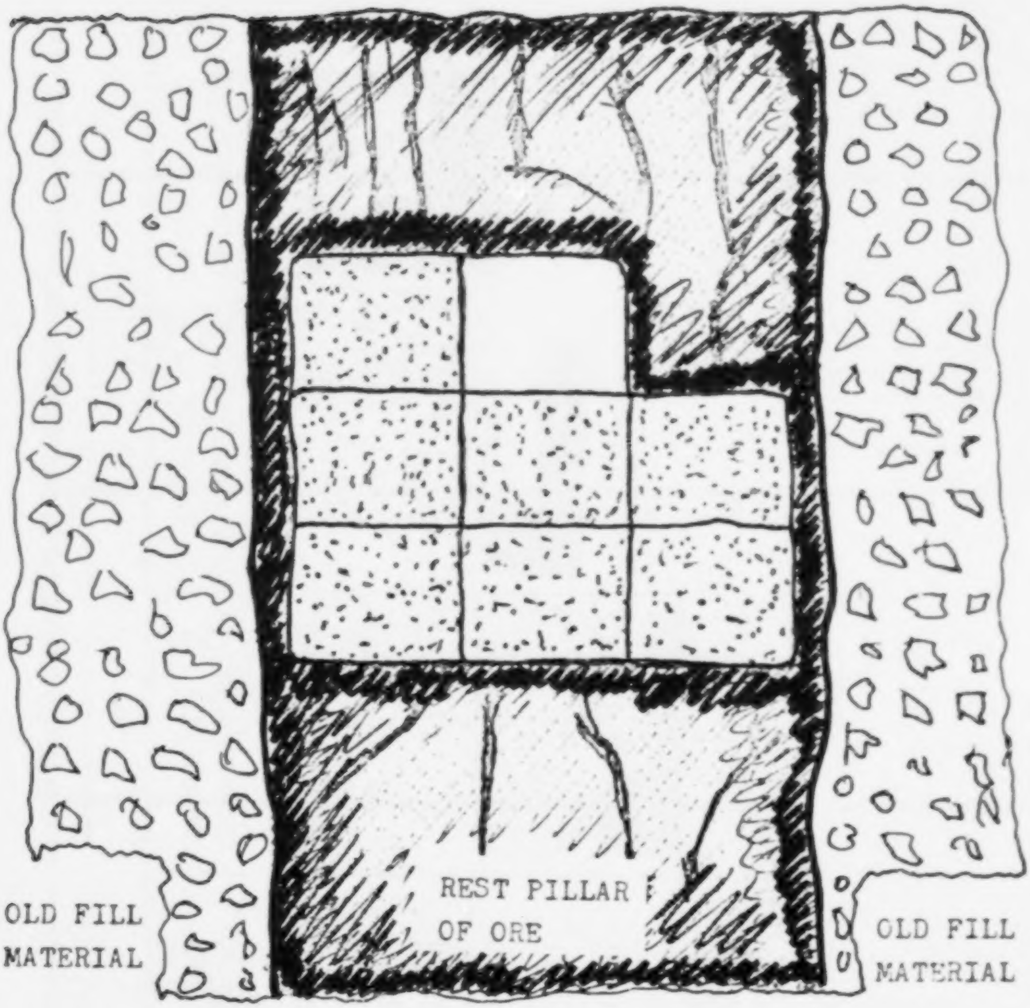
OLD FILL MATERIAL
REST PILLAR OF ORE
OLD FILL MATERIAL

Figure 67

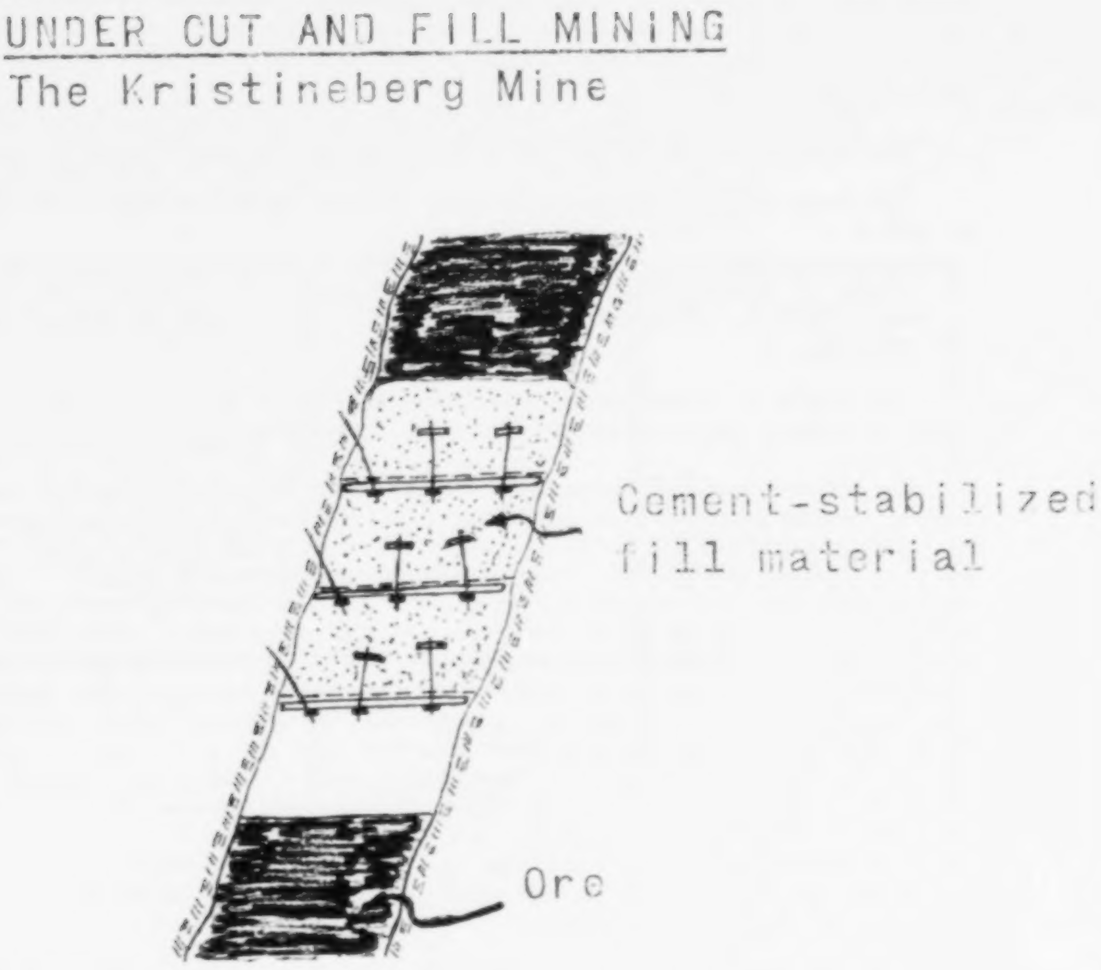
UNDER CUT AND FILL MINING
The Kristineberg Mine
Cement-stabilized fill material
Ore

Figure 68. Undercut-and-fill: Kristineberg mine

the gravity flow took place along rather slanting sides, partly in order to reduce the length of the drill holes. However, recent investigations have shown that the volume of motion is very high and narrow, and therefore it has been suggested that drilling should be carried out with more or less vertical and parallel holes.

In figure 69 the fundamental parameters have been defined.

Dividing layer: an imaginary layer between material in motion and material at rest.

Angle of dividing layer: the angle between the dividing layer and the horizontal plane. This angle is determined by the angle of inner friction of the material.

The angle of inner friction is determined by: particle size distribution, particle shape, hardness of the material, cohesion, degree of packing, etc.

In order to investigate, analyse and solve disadvantages with sublevel caving, rather extensive calculations have been made in recent years, as well as model and full-scale tests. A full-scale test has been carried out in a mine in central Sweden and simultaneously a model-scale test at 1 in 20 was done in laboratory. The intention with these two tests was twofold. The actual gravity flow in broken ore has, as far as we know, never been investigated in a precise and controlled way and we also wanted a real comparison between full-scale and model tests with the same geometry. The full-scale test comprised three sublevels with constant parameters during the whole test. About 13,000 numbered markers were inserted in 9,000 m of drill holes between the ordinary blast hole fans. Blast holes were loaded with a charge density of 0.7-0.75 kg/m^3. The ignition was effected with VA-detonators. The mucking was done by a loading machine LM 30 to a dumper U2N of 2.5 ton. While loading, all recovered markers are collected and registered. Moreover, the blasted rounds are weighed, screened and photographed in order to be able to compare results from different rounds.

The results from the tests show that the gravity flow in blasted rock is different from what has been assumed. The flow ellipsoid is considerably thinner and longer, which necessitates an adjustment of drift locations and sublevel heights in comparison with prior methods (figure 70).

A comparison of the results from full-scale tests and model tests shows that the shapes of the volumes of motion differ somewhat. In the model tests, the volume of motion is rather narrower and higher and more evenly shaped. This is natural in view of the more evenly sized and more easily moving material in the model. The angles of inner friction have been measured to about 35° in the model and 45-50° in normal blasted rock in the mine. In addition, loosening-up and density of the material are more favourable for an even gravity flow in the model tests.

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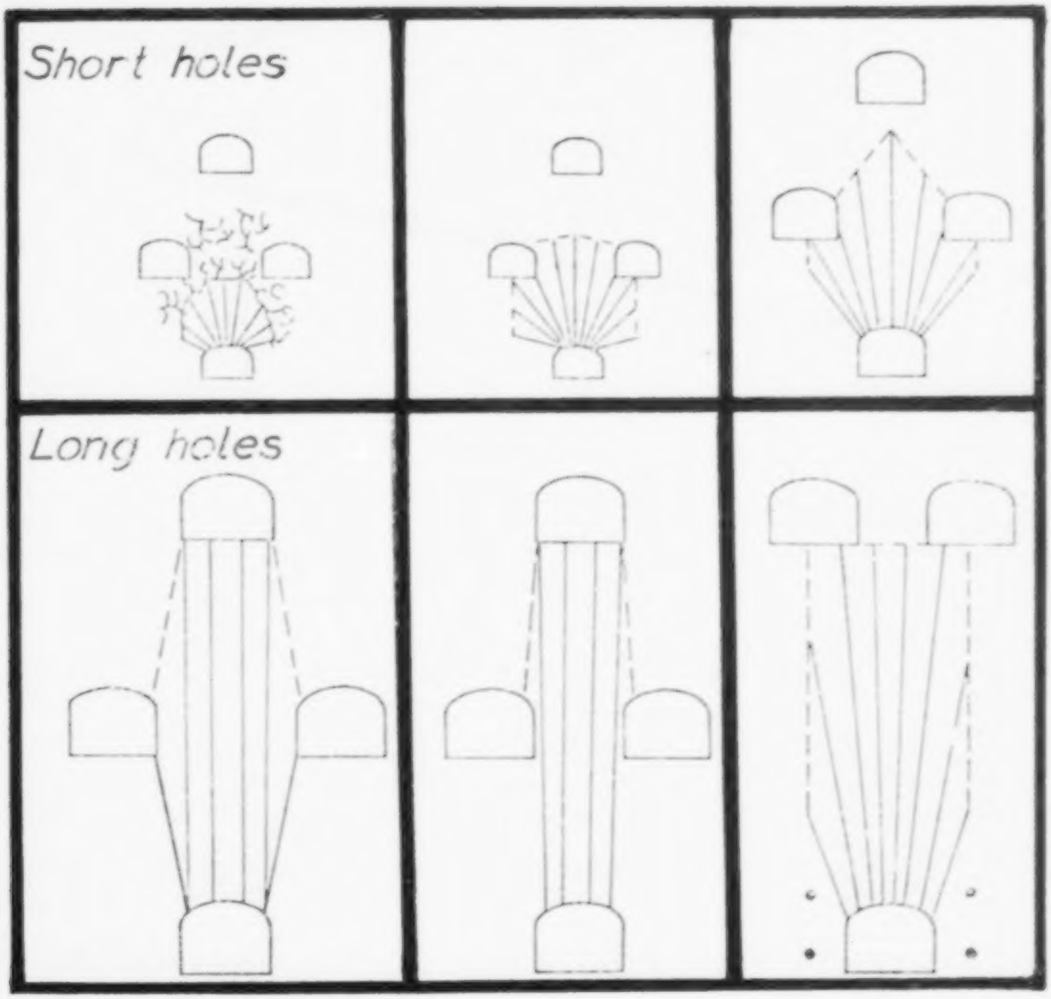
Development of the sub-level caving method
Short holes
Long holes

Figure 69

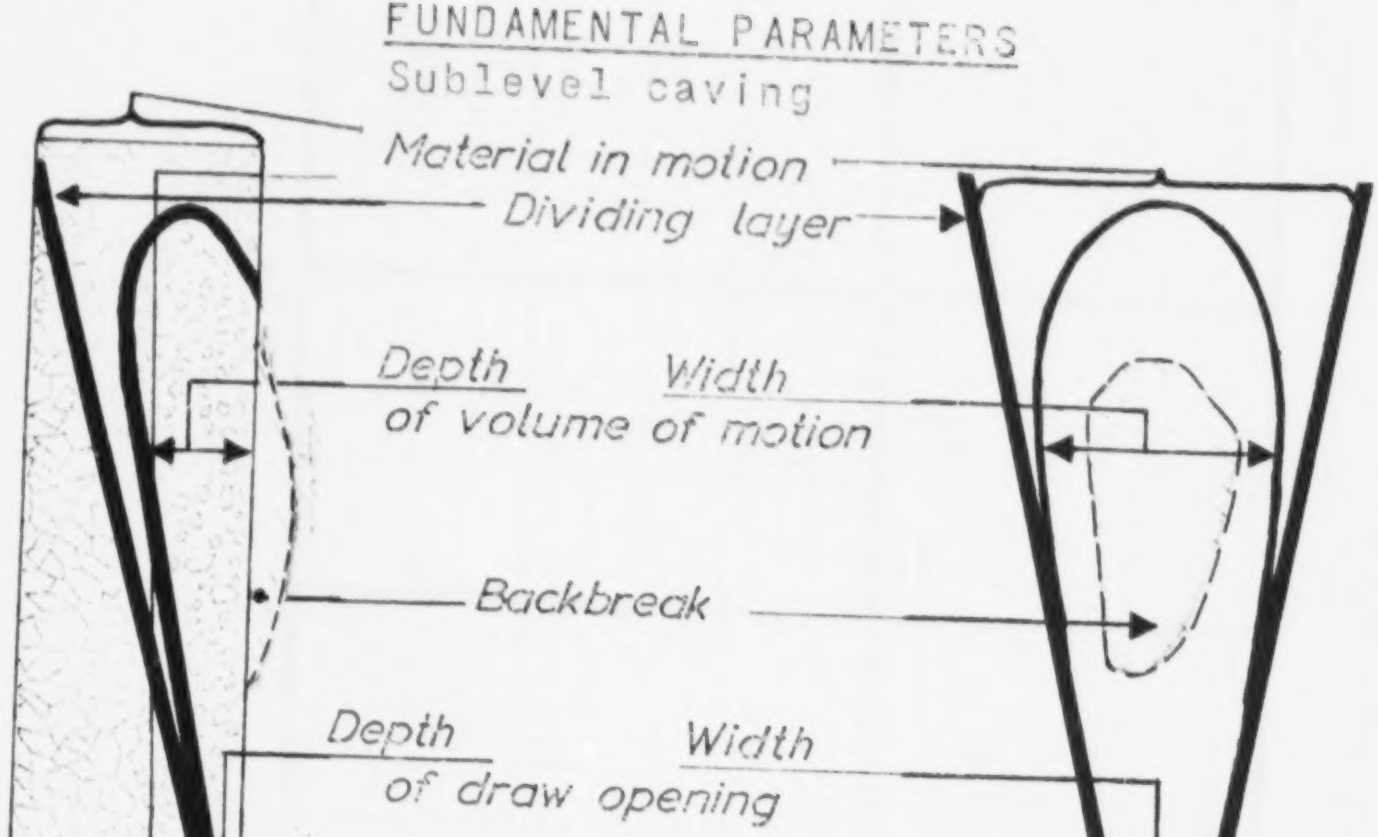
FUNDAMENTAL PARAMETERS
Sublevel caving
Material in motion
Dividing layer
Depth Width
of volume of motion
Backbreak
Depth Width
of draw opening

Figure 70

A proof for both sensitivity and accordance between model tests and full-scale tests is the pronounced tendency of inclination of the volumes of motion towards the place where the waste-rock was introduced.

There is a pronounced accordance between the absolute height of the volumes of motion and the distance between the apex and the mining front at 90° inclination, as well as between the absolute height and the horizontal extension of the volumes of motion.

Figure 71 shows a considerable backbreak, volume-wise, into the solid, remaining front. This was rather surprising and has probably not been studied before. Backbreak is of great disadvantage since this space is filled with waste rock for each round, and this waste rock is added to the volume of the round.

Figure 72 is a diagram of the final results from full-scale and model tests. The deviation of values for each curve is surprisingly small. The diagram shows that the height of the volumes of motion increases rapidly in the beginning, and already after loading 50 per cent of the theoretical quantity, the height is over 15 m, which should be compared with 18 m for 100 per cent degree of loading.

In sublevel caving, the cost of drifting is a very large part of the total costs, and it is therefore desirable to produce as many tons per meter of drift as possible. The table shows the theoretical quantity of ore per meter of drift for different drift dimensions and sublevel heights.

As an example some mining layouts for longitudinal sublevel caving are shown. If the longitudinal drift is located in ore adjacent to the footwall, as in figure 73, large orelosses will occur along the footwall, and if waste rock of fine fragmentation flows down, the ore on the footwall will be almost completely lost.

If, on the other hand, the longitudinal drift is located partly into the footwall as in figure 74, a certain amount of the footwall ore will be recovered, and this reduces the orelosses.

It is also an important point for transverse sublevel caving where the longitudinal drift should be located in relation to the footwall. The most economic location depends on the dip, sublevel height and drift dimensions.

Unfortunately, sublevel caving is too complicated to allow a specification or even a generalized recommendation of definite parameters.

About 10 years ago, the idea of front caving was put forward. Figure 75 shows drill patterns for normal sublevel caving and front caving for given dimensions of drifts and pillars. As the figure shows, front caving has twice the sublevel height compared with normal sublevel caving, but even so the maximum drill-hole lengths are the same.

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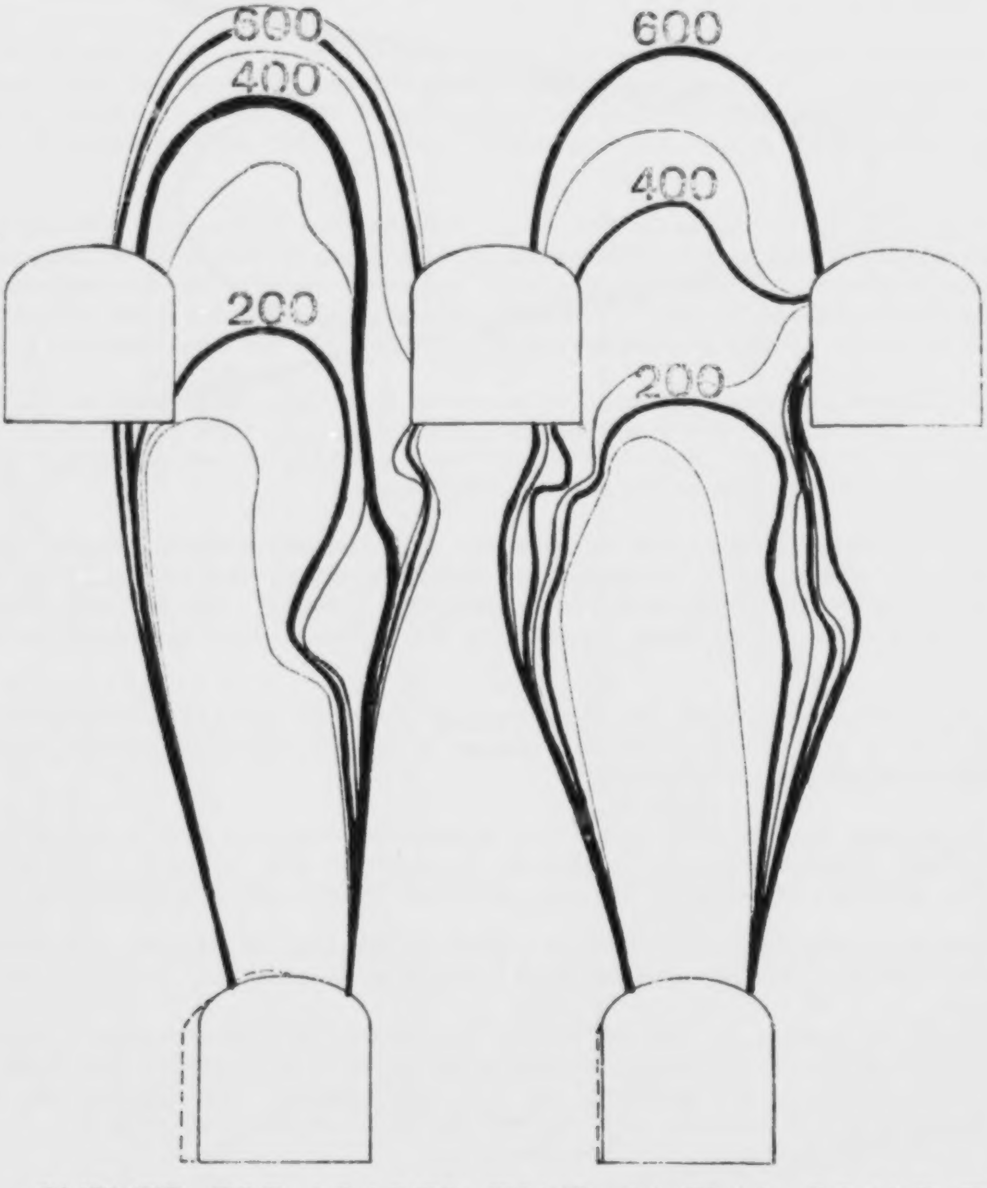
FULL SCALE TEST OF SUBLEVEL CAVING

VERTICAL SECTION PERPENDICULAR TO SUB LEVEL DRIFTS



CURVES FOR LOADED TONS IN FULL SCALE TEST
(Manually designed for marking plane 45 cm in front of mining front)

Figure 71. Volumes of motion at sublevel caving

Example of back break after blasting a round in sub-level caving

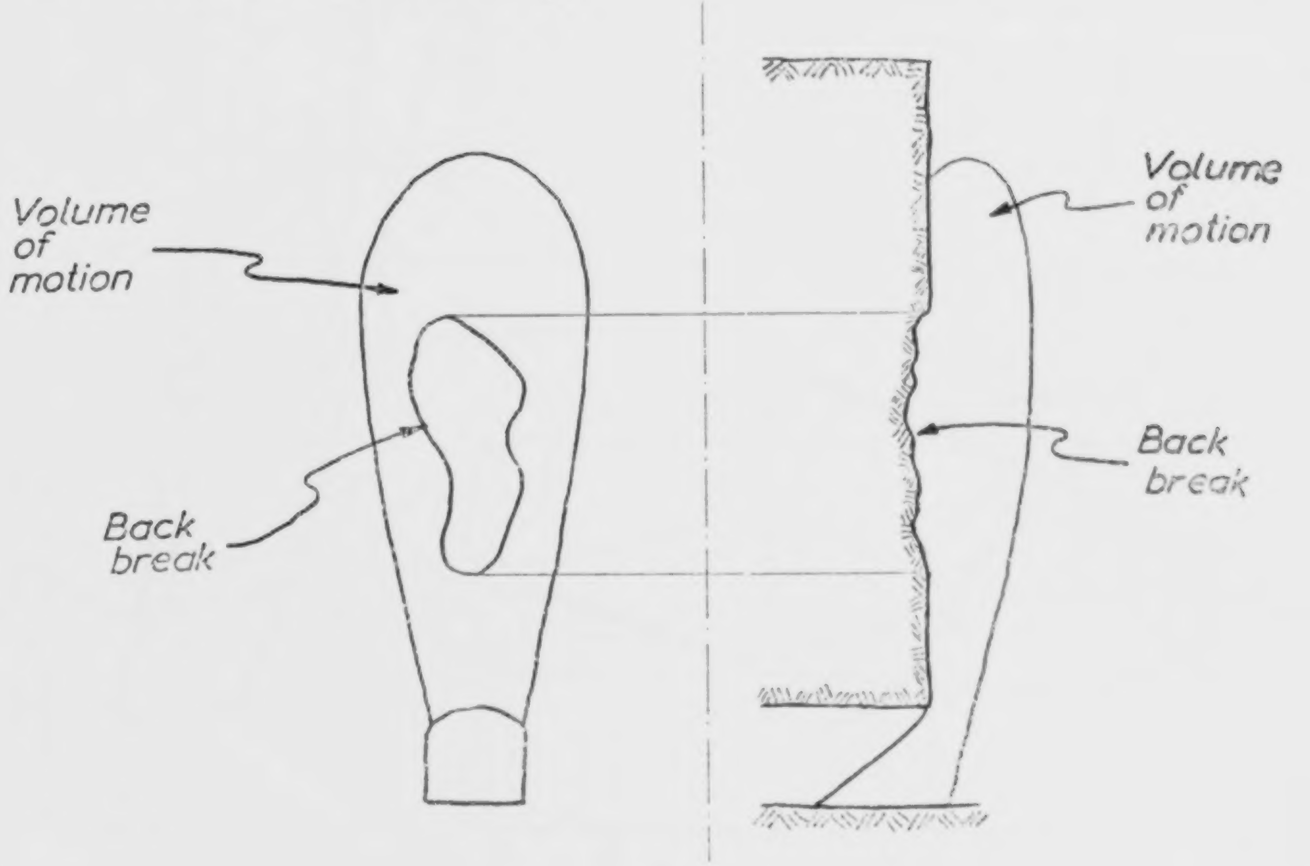

Volume
of
motion
Back
break
Volume
of
motion
Back
break

Figure 72

MAXIMUM HEIGHT, WIDTH AND DEPTH OF VOLUMES OF MOTION: FROM FULL SCALE AND MODEL TESTS.

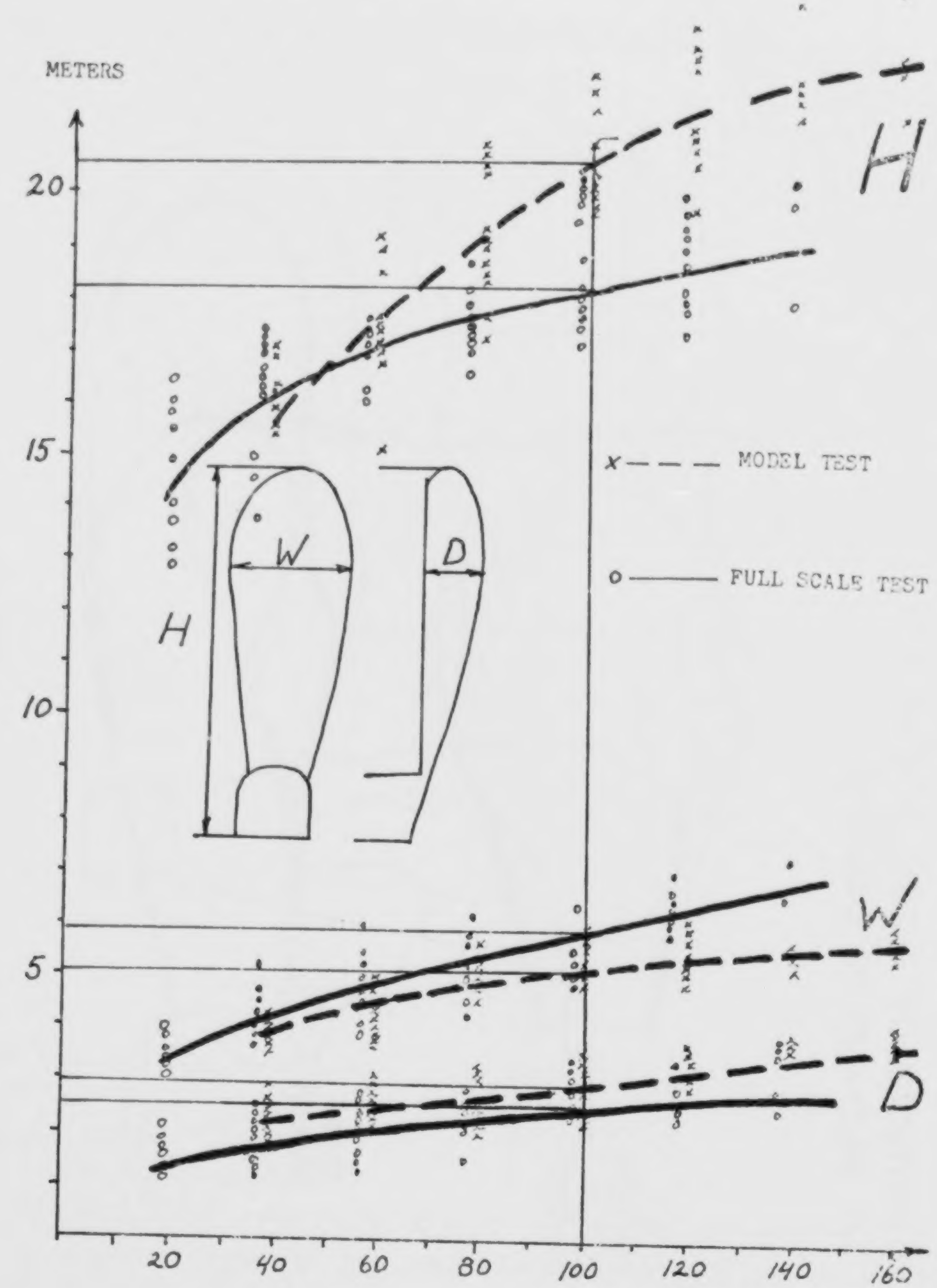


Figure 73. Gravity flow curves

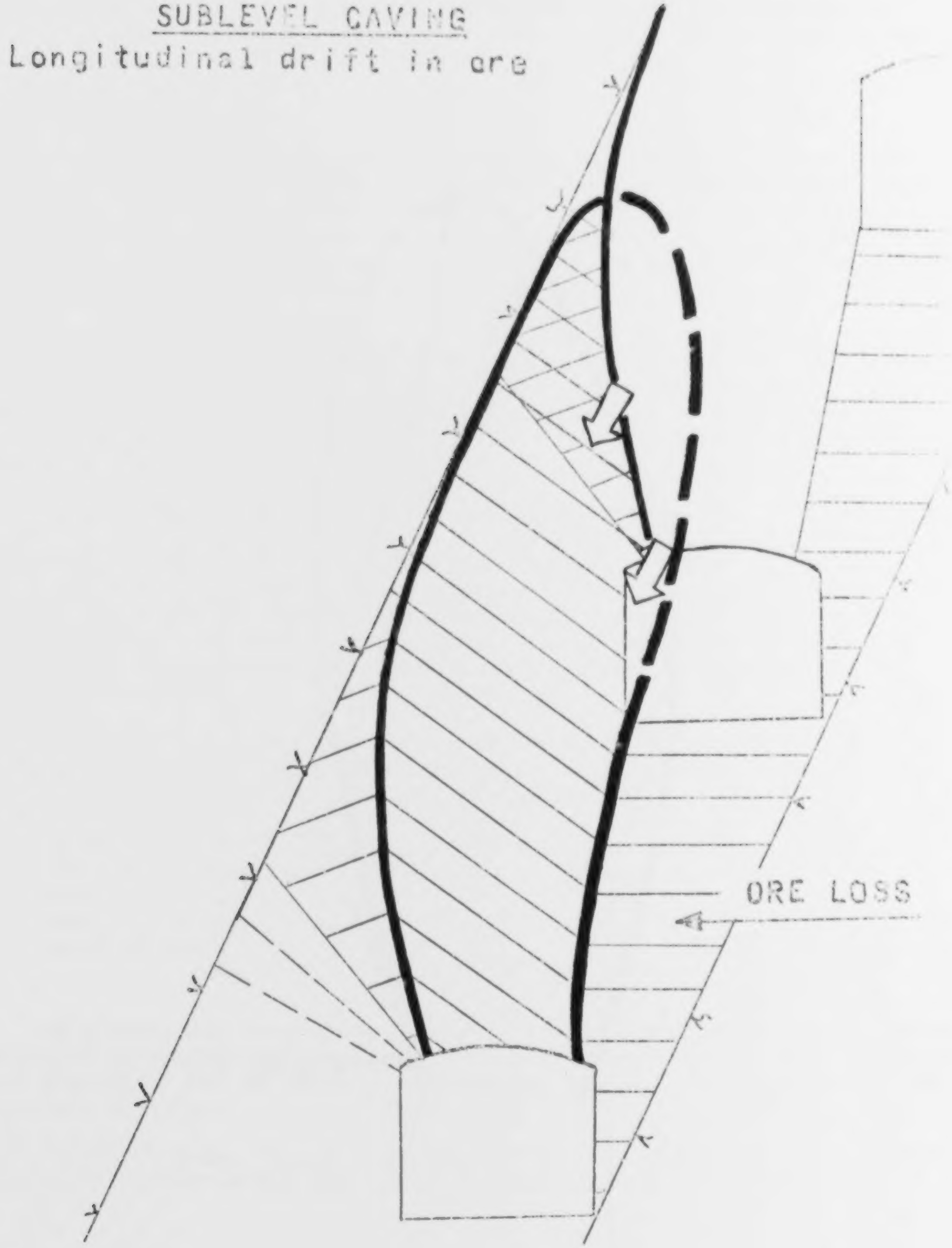

SUBLEVEL CAVING
Longitudinal drift in ore
ORE LOSS

Figure 74. Longitudinal mining: drift situated in ore

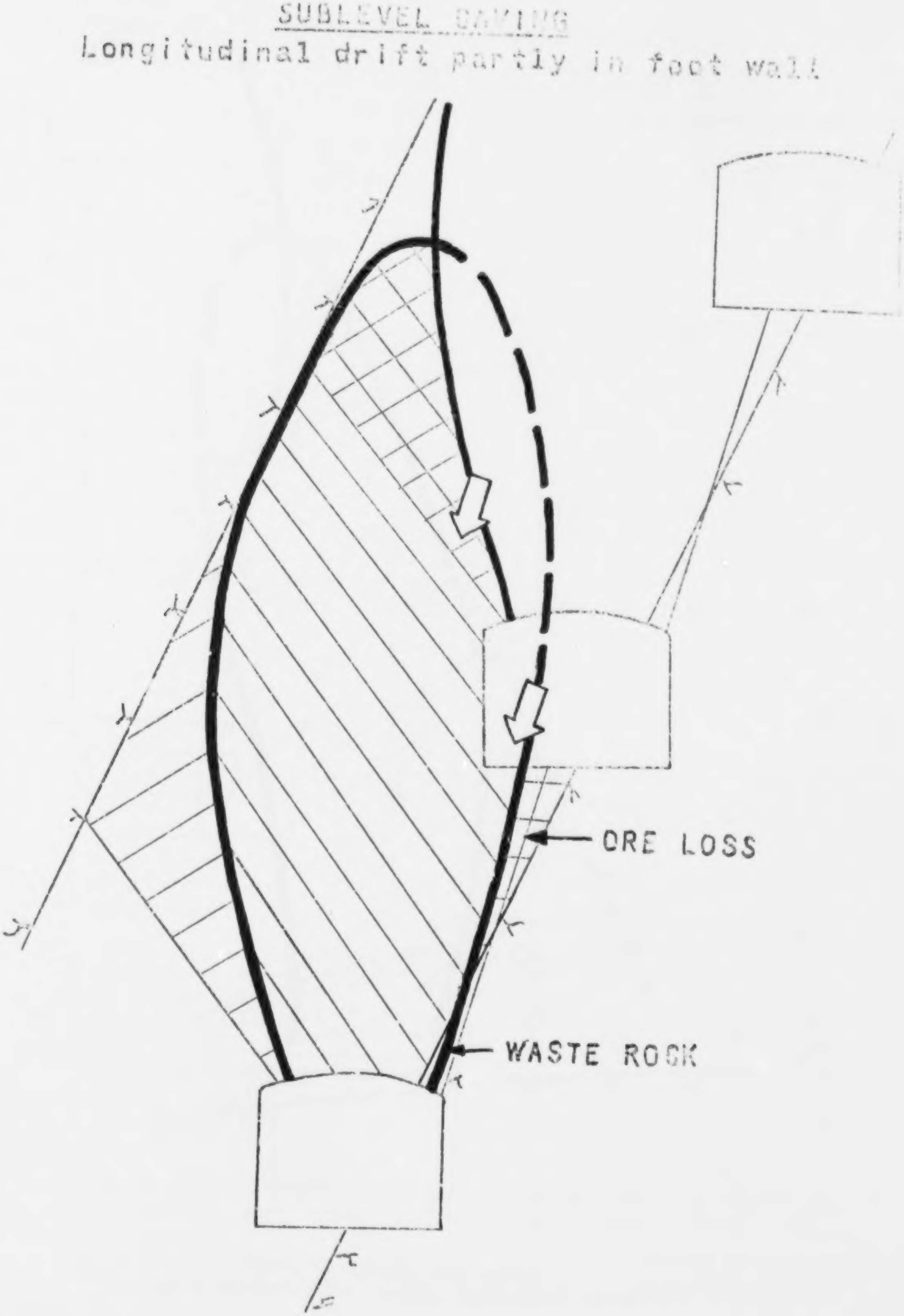

SUBLEVEL CAVING
Longitudinal drift partly in foot wall
ORE LOSS
WASTE ROCK

Figure 75. Longitudinal mining: drift situated partly in foot-wall

A comparison of normal development work for front caving and sublevel caving shows that considerably less drifting is required for front caving, both in ore and waste rock. Because of large sublevel height, front caving cannot be applied in narrow or flatly dipping orebodies.

Block caving

Block caving is a mining method whereby a large block is undercut so that caving is induced in overlying ore. Destruction causes fragmentation which allows the ore to flow through crater openings of a certain size down to a loading level. Normally the method is applied only to large orebodies of low strength, having a fine fragmentation and under the conditions that the ore is of low value and mining is carried out to a certain cut-off grade.

In Sweden, block caving is at present used only at Grängesberg. The principle layout is shown in figure 76. Loading is carried out from craters, opening out into the walls of crosscuts. For loading and transport, rocker shovel loaders LM 100 and 2 m^3 Granby shuttle cars are used. At present, loading and transport with diesel driven CAVO D 710 loaders is also being tested with good results.

Chile is one of the countries where block caving is used on a very large scale. About 35 million tons are mined annually with this method. Figure 77 shows the principle layout at El Teniente within that part of the orebody, where the ore is weak and has a fine fragmentation. Secondary blasting is insignificant and the capacity per drawpoint with 1.5 x 1.5 m opening is 300 tons/shift. In those parts of the very large orebody where the ore is harder, the fragmentation gets coarser and the capacity may be reduced to about 30 tons/shift.

According to an article published at the World Mining Congress in Bucharest, September 1972, underground mining of iron ore in Russia has the following distribution of mining methods (percentages):

Block caving	51.7
Induced block caving	11.7
Room and pillar	20.9
Sublevel caving	14.0
Other methods	1.7
	100.0

An interesting observation is that many of the articles on mining, published in Russia during the last 10 years, deal with induced caving, that is drilling and blasting of the ore before it reaches the loading level, in order to reduce secondary blasting.

Figure 78 shows "induced block caving" in 1964, where the ore is blasted against an already broken mass of ore by means of long blastholes.

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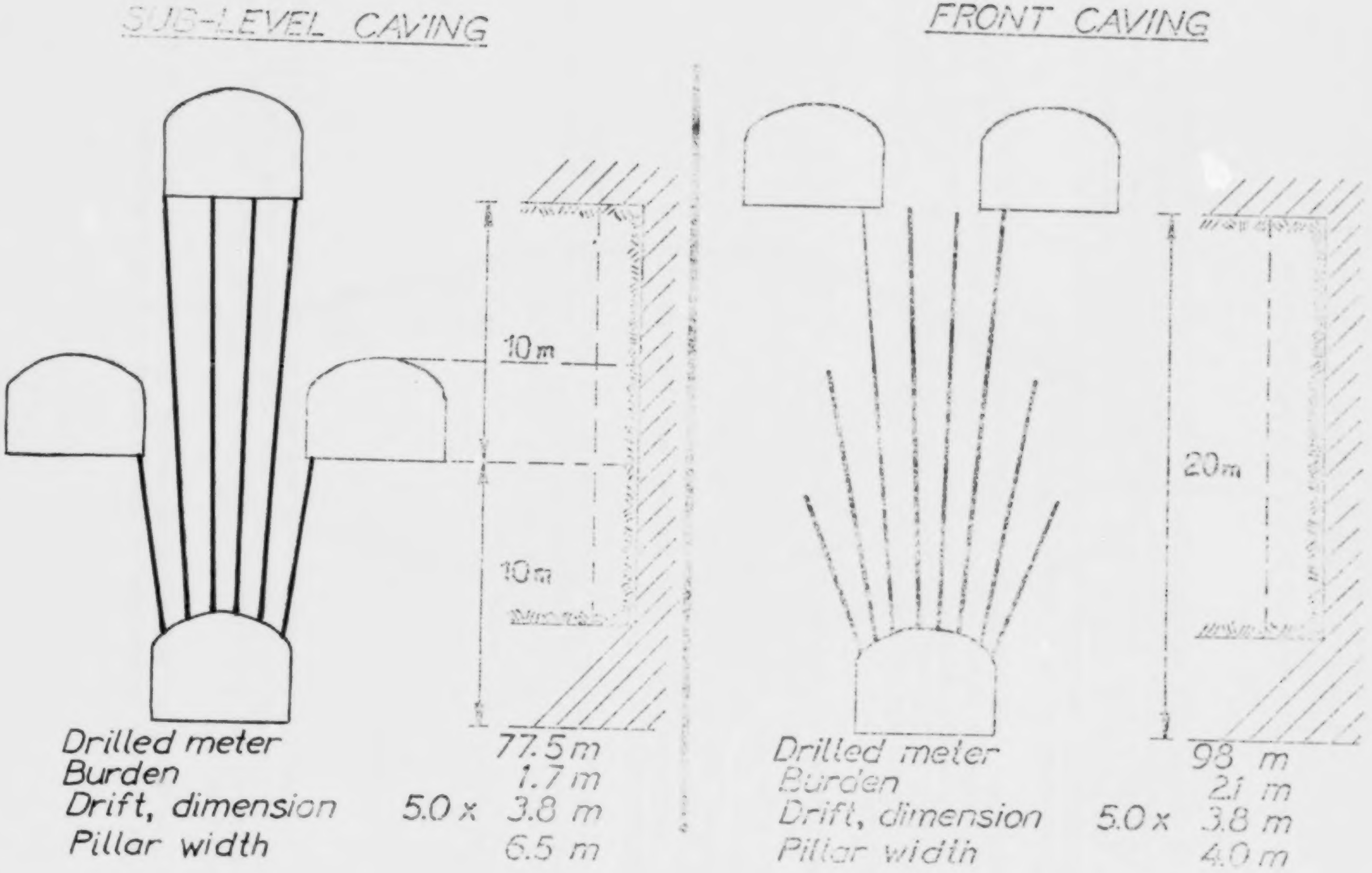
SUB-LEVEL CAVING
FRONT CAVING
10 m
10 m
20m
Drilled meter 77.5 m
Burden 1.7 m
Drift, dimension 5.0 x 3.8 m
Pillar width 6.5 m
Drilled meter 98 m
Burden 2.1 m
Drift, dimension 5.0 x 3.8 m
Pillar width 4.0 m

Figure 76. Drilling pattern, comparing sublevel caving and front caving

BLOCK CAVING, GRÄNGESBERG

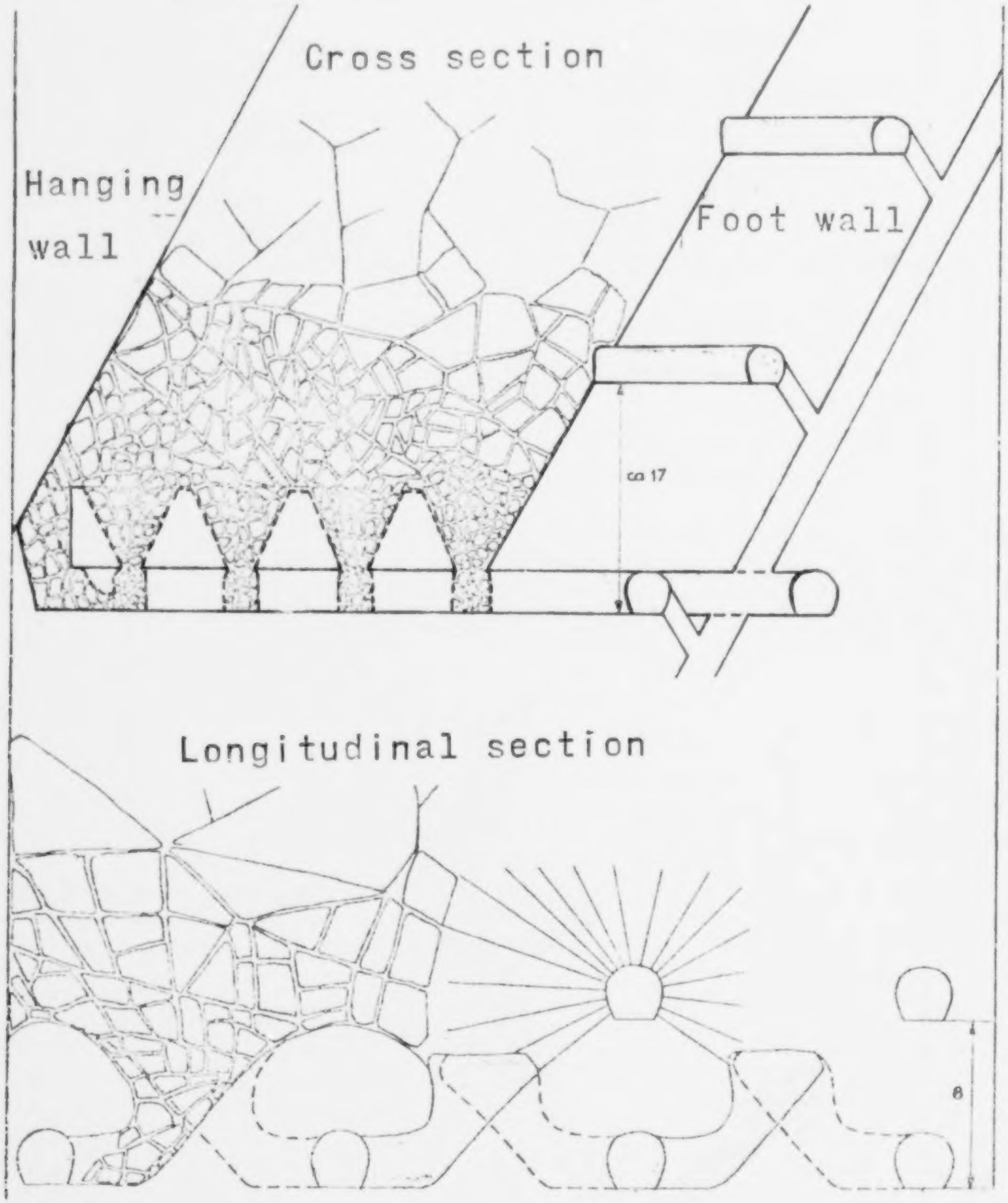
Cross section
Hanging wall
Foot wall
ca 17
Longitudinal section
8

Figure 77

BLOCK CAVING, EL TENIENTE

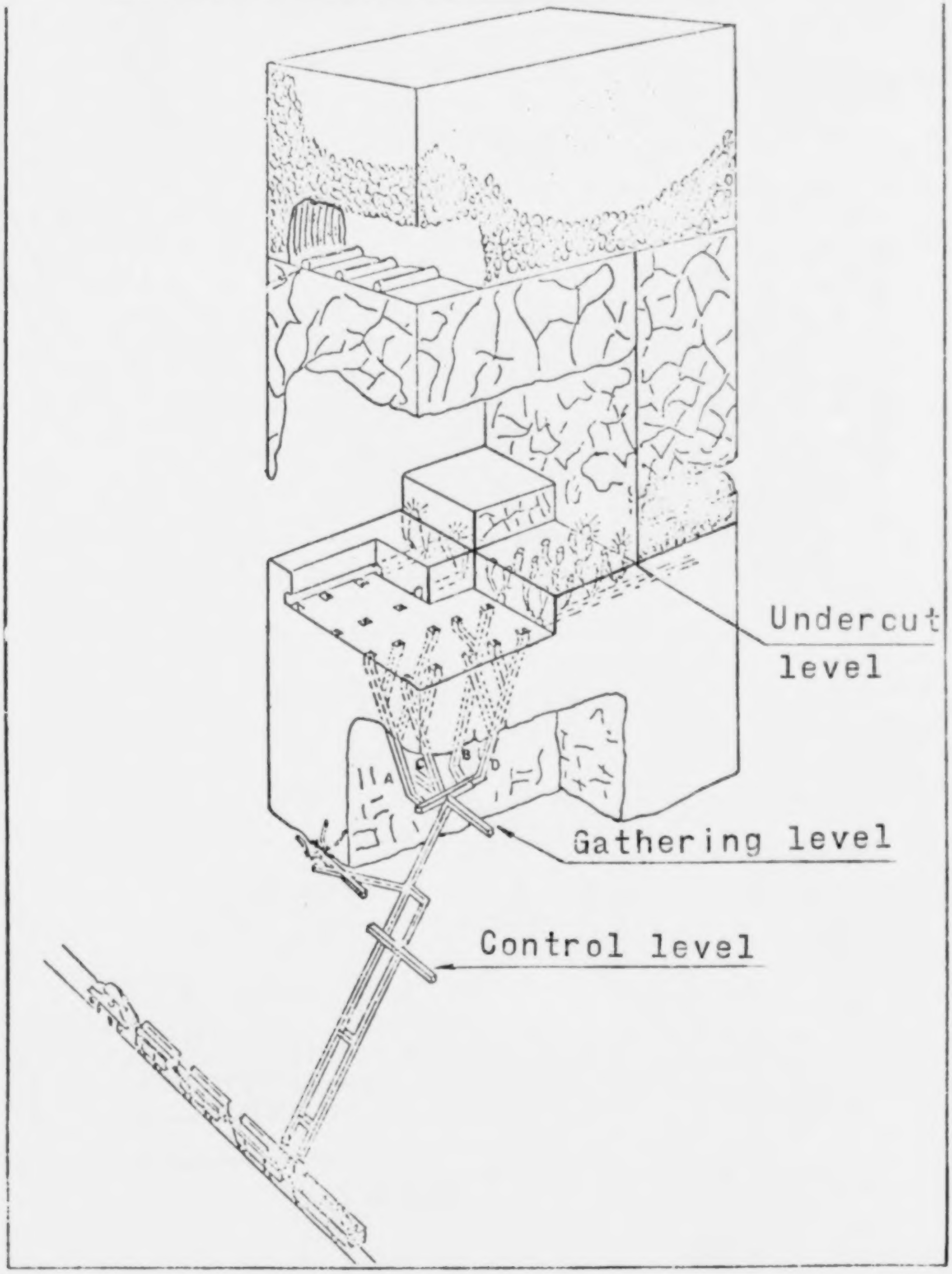
Undercut level
A
C
B
D
Gathering level
Control level

Figure 78

Roughly the same principle is used according to an article, published in 1972. The following data have been given:

Drill-hole diameter	100-150 mm
Drill-hole length	15- 30 m
Specific charge	0.300-0.800 kg/ton
Number of rows blasted simultaneously (one section 15-25 m)	7-10 rows
Boulder frequency	4-10%
Swelling during blasting against already broken ore	Depends on specific charge, etc.
Degree of loosening up of already broken ore	1.08-1.5 (1.3 desired)
Orelosses	13-16%
Waste rock dilution	About 10%

It is stated in the article that a better fragmentation of the ore is obtained if blasting takes place against densely packed rock.

The principle of making a slot along the footwall has been used in several mines, for example, in Zambia and at INCO's Creighton Mine in Canada. Figure 79 shows a cross section of Creighton Mine, which contains partly low grade nickel ore. Loading is carried out to a certain cut-off grade. See also figures 80 and 81.

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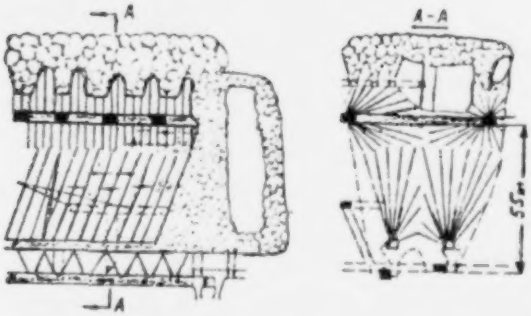
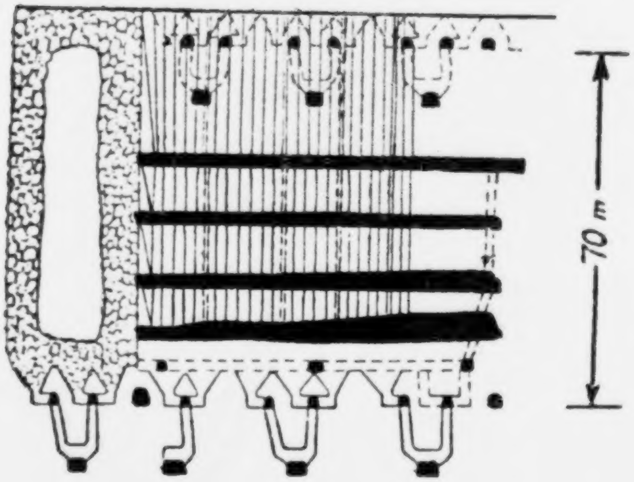


Drilling and blasting against already caved ore according to Imenitov 1964



Drilling and blasting against already caved ore according to Stebakor 1972

Figure 79. Induced block caving, Russia

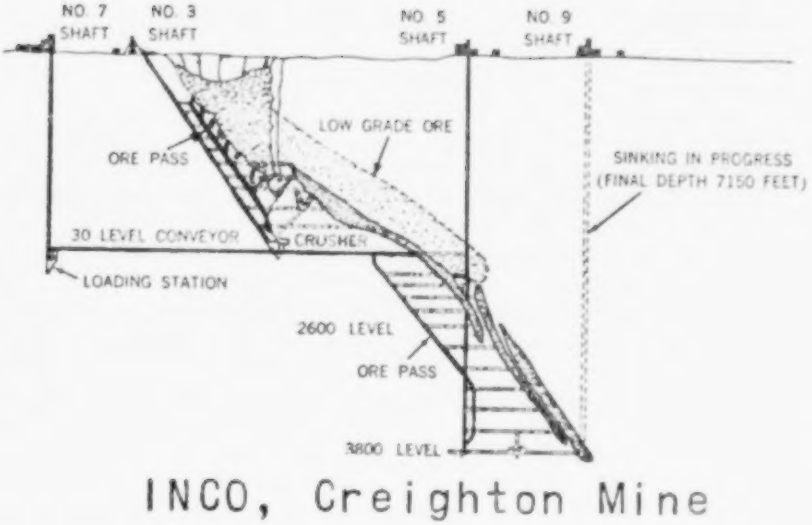

NO. 7 SHAFT
NO. 3 SHAFT
NO. 5 SHAFT
NO. 9 SHAFT
LOW GRADE ORE
ORE PASS
SINKING IN PROGRESS (FINAL DEPTH 7150 FEET)
30 LEVEL CONVEYOR
CRUSHER
LOADING STATION
2600 LEVEL
ORE PASS
3800 LEVEL
INCO, Creighton Mine

Figure 80. Block caving, Creighton Mine

VENTILATION ALTERNATIVES

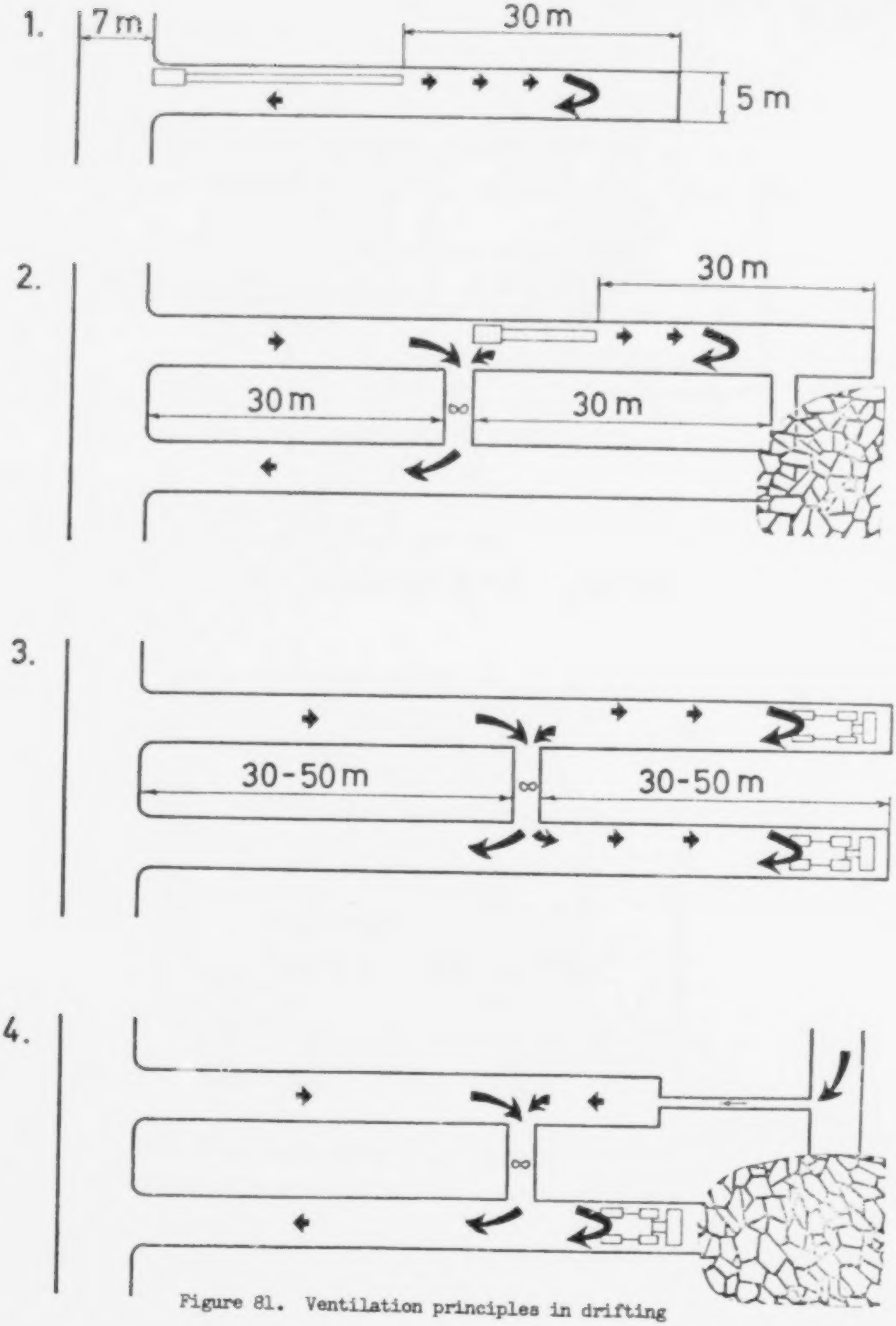

1.
7 m
30 m
5 m
2.
30 m
30 m
∞
30 m
3.
30-50 m
∞
30-50 m
4.
∞

Figure 81. Ventilation principles in drifting

VII. SINKING THE CREIGHTON NO. 9 SHAFT AT SUDBURY

W. R. Dengler*

In the early fall of 1965, a contract was let by the International Nickel Company of Canada Ltd. to Shaft Sinkers (Canada) Ltd./J. S. Redpath Ltd. Joint Venture, to sink and equip the Creighton No. 9 Shaft at their Creighton Mine property in Sudbury, Ontario, Canada. This paper describes the conditions encountered, the sinking method used, and the results obtained. It is hoped that presenting some basic performance figures in conjunction with the conditions under which they were obtained will assist those using this information to plan similar shafts in this part of the world.

A. The shaft

The shaft, which is circular in section, is lined with monolithic concrete, a minimum of 12 inches. Its inside diameter is 21 feet, and steel sets are installed on 15-foot centres. The shaft depth is 7,137 feet. This is the deepest shaft of its kind sunk in the Western Hemisphere.

An idea of the quantities involved on this project are shown in the following figures:

Shaft excavation: 3,370,000 cubic feet equivalent to 10 miles of 8' x 8' drift

Stations excavated, rock-bolted, and concreted during sinking: 422,212 cubic feet equivalent to 6,600 feet of 8' x 8' drift

Rock-bolts installed: 264,314 linear feet (laid end to end, these bolts would stretch 50 miles)

Screen installed: 415,742 square feet

Concrete in shaft lining and stations: 40,000 cubic yards (enough concrete to lay 40 miles of sidewalk)

Steel used in equipping the shaft: 6,000,000 pounds

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* General Manager, J. S. Redpath Ltd. North Bay, Ontario, Canada.

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The equipped shaft has two skip compartments, a cage compartment, a counter-weight compartment, a utility compartment and a manway. The manway compartment and the skip compartments are enclosed by chain link brattice. The skips and counter-weight run on rectangular hollow section steel guides and the cage runs on Fir guides backed by steel wide-flange beams. Figures 82 and 83 show the general arrangement and plan of the shaft.

The finished shaft utilizes 15-ton skips which are hoisted at 3,300 feet per minute from the 6,600 loading pocket. The cage hoists up to 96 men per trip. The shaft has electrical junction stations, cut at 750 foot intervals to a depth of 3,850 feet below surface. From this point on, regular stations are located every 200 feet down to the 7,000 foot elevation.

Loading pocket complexes are cut at the 5,200 foot elevation, the 6,600 foot elevation and at the 7,000 foot elevation. Each complex consists of a loading pocket-conveyor way, chair pocket, and spill pocket.

To summarize the shaft excavations, there are: 5 electric junction stations; 16 shaft stations; 3 loading pockets; 3 chair pockets; 3 spill pockets; and a total of 30 openings in the shaft lining.

B. Ground conditions

The sinking took place through Norite down to between 3,000 and 4,000 feet. This is fairly hard drilling ground and tends to give poor fragmentation when blasted. The large fragments are hard on equipment and large mucking equipment was required to handle this rock effectively. Equipment damage from blasting and difficulty in mucking are directly proportional to the size of the fragments.

Little rock-bolting was required in the Norite and, in general, ground conditions were good. We were able to keep our concrete lining from 30 to 45 feet from the shaft bottom. Conditions changed below the 3,850 level. A mixture of gabbro and granite was encountered and the balance of the shaft was sunk in this ground. At this point, inherent stress in the rock became evident and some spallin or minor bursting took place. It became necessary to rock-bolt and screen all the exposed rock and to carry the concrete lining a maximum of 18 feet from the shaft bottom. This meant that when the 15-foot forms were lowered, the concrete was brought to within 3 feet of the shaft bottom and resulted in more damage to shaft forms as well as imposing restrictions on mucking, drilling and forming owing to the loss of working room.

To ensure safety during the sinking operation, it became necessary to rock-bolt and screen during the mucking cycle such that a maximum of only six feet of open wall was exposed. Destressing was also carried out by drilling three 16-foot long holes ahead of each bench and blasting them with the bench.

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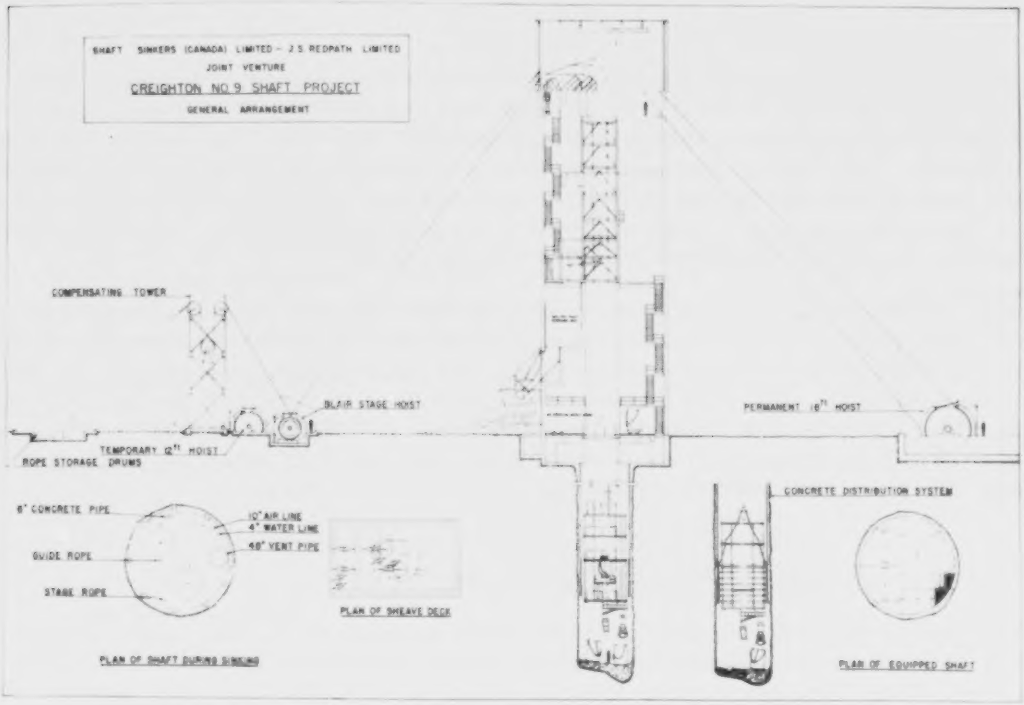
SHAFT SINKERS (CANADA) LIMITED – J.S. REDPATH LIMITED
JOINT VENTURE
CREIGHTON NO 9 SHAFT PROJECT
GENERAL ARRANGEMENT
COMPENSATING TOWER
BLAIR STAGE HOIST
TEMPORARY 12 FT HOIST
ROPE STORAGE DRUMS
PERMANENT 18 FT HOIST
6" CONCRETE PIPE
10" AIR LINE
4" WATER LINE
48" VENT PIPE
GUIDE ROPE
STAGE ROPE
PLAN OF SHAFT DURING SINKING
PLAN OF SHEAVE DECK
CONCRETE DISTRIBUTION SYSTEM
PLAN OF EQUIPPED SHAFT

Figure 82. General arrangement of Creighton No. 9 shaft

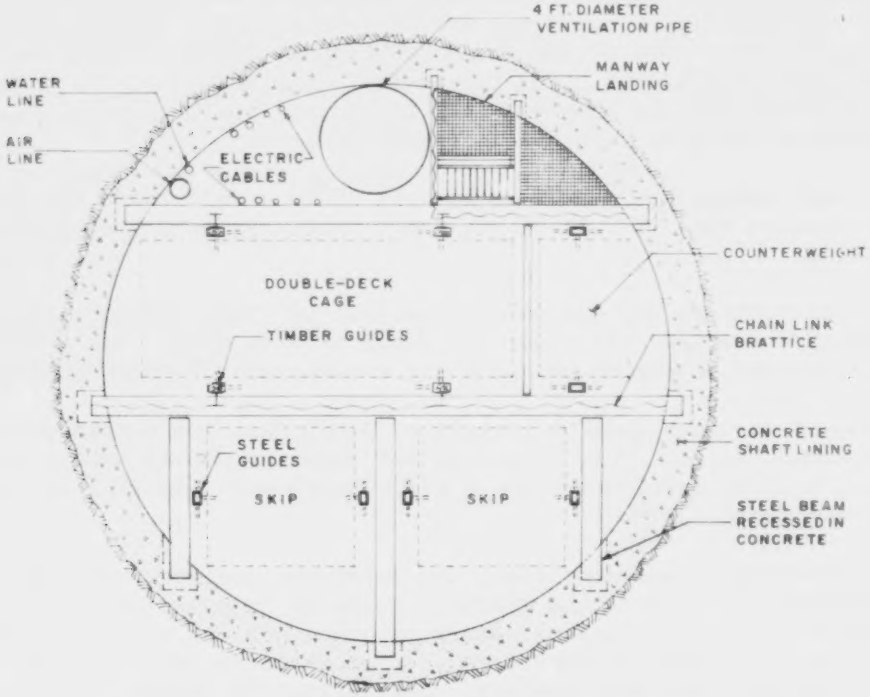
4 FT. DIAMETER
VENTILATION PIPE
MANWAY
LANDING
WATER
LINE
AIR
LINE
ELECTRIC
CABLES
COUNTERWEIGHT
DOUBLE-DECK
CAGE
TIMBER GUIDES
CHAIN LINK
BRATTICE
CONCRETE
SHAFT LINING
STEEL
GUIDES
SKIP
SKIP
STEEL BEAM
RECESSED IN
CONCRETE

Figure 83. Plan of Creighton No. 9 shaft

For overbreak control, perimeter powder was being used. However, when bursting walls gave us trouble, we changed our blasting practice and loaded the perimeter holes completely with high explosives. The bursting and the overbreak were reduced. The theory here is that the perimeter powder tended to create a near-perfect circle which was strong in section and built up stresses until it finally burst violently. Using the solid columns of high explosives, we think the shattering effect served to relieve the pressure.

When cutting stations, the crew drilled two 50-foot long destress holes and blasted them with the first round. We also had to rock-bolt and screen before mucking and during mucking. We were unable to work under open ground at any time for safety reasons. It is of interest to note that during the period that the most difficult ground conditions were encountered, the best safety record was established and in fact, the whole crew went for one year with only one lost-time accident due to a rash caused by contact with wet concrete.

C. The sinking method

The shaft was sunk using a galoway stage suspended on six parts of rope and operated by a Blair stage hoist. Excavation by the benching system advanced the shaft about four feet per bench. It is believed that the benching method is desirable in ground where bursting may be a problem as it destresses the bottom. Bottom bursting was definitely less common in this shaft than in a similar shaft of smaller diameter sunk in the same formation using the full-face method. For those contemplating the use of full-face methods in highly stressed ground, it should be noted that difficulty was encountered during a full-face trial at a depth of 5,000 feet. The bottom would continue to spall during cleaning until the centre was five or six feet below the walls. This made cleaning a long process, and if the bottom was not cleaned until it stopped spalling, the drillers would end up drilling in loose rock - a difficult procedure.

The shaft lining was poured in 15-foot lifts as the shaft was sunk. Utility lines for sinking were carried on the shaft wall. Permanent steel work was installe after sinking.

When the contract was awarded in the fall of 1965, the collar had been excavate and partially concreted. No other buildings were on site. Six months later sinking started, using the permanent headgear and a temporary 12-foot sinking hoist.

The shaft was then sunk to a depth of 3,850 feet and a change-over was made to the permanent hoist. This is an 18-foot diameter direct drive, double drum Winder with a 6,500 hp DC motor and a 7,000 hp MG set. Sinking and station-cutting were then completed.

When the sinking was completed, the permanent headgear was completed and dry-up grouting was done in the shaft. It should be noted here that owing to the sinking configuration, it was necessary to leave out the headframe steel work and one sheave deck to sink the shaft. A four-month strike at Inco also took place at this time, and the crew was unable to work during the strike.

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Equipping then started and was completed early in January 1971. The following is a summary of how the time was utilized from the day the contract was let up to the completion of equipping.

Original set-up, hoist change-over at the 3,850 level and headframe completion and set-up for equipping (it should be noted here that drywall grouting was done concurrently with the headframe completion)	296
Strikes at Inco (two)	130
Grouting during sinking	48
Days not worked (holidays)	42
Hoist maintenance and delays (One shift/week plus actual delays)	73
Station-cutting and concreting, rock-bolting and screening of the stations	313
Sinking and contractor's delays	726
Equipment, including the loading pocket, spill pocket, and chair pocket installation, and installation of nine electrical cables and the signal system in the shaft	302
Total	1,930 calendar days

Dividing this total by the shaft depth an advance/calendar day since the contract was awarded . . . . 3.7 feet/day

The following are now subtracted from the total:

Set-up and change-over, etc.	296
Strikes	130
Grouting	48
Station-cutting	313
Total	787
Balance	1,143 days

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Average advance per calendar day sinking, including all delays, holidays, ground support, and equipping, except as listed above . . . . 6.2 feet/day

Station-cutting including concreting, rock-bolting and screening was:

$$\frac{422,212}{313} = 1,350 \text{ cubic feet/calendar day}$$

Statistics comparing with the foregoing figures are:

(a) The average advance/day from surface to the 3,850 level during sinking, including contractor's delays was 11.2 feet/day, and the best month was 411 feet of shaft sunk and lined;

(b) The average advance/day from the 3,850 level to the bottom was 8.3 feet/day;

(c) Installation of steel sets including contractor delays takes place at a rate of 50 feet/day but when electric cables, loading pockets, chair pockets, spill pockets, etc. are added, the figure is reduced to 23.7 feet/day;

(d) The average sinking rate including holidays and all delays other than grouting and strikes was 8.5 feet/calendar day from surface to the shaft bottom. This figure does not include equipping.

D. Summary

From the data presented, several points should stand out:

(a) Set-ups, change-overs, etc., can be extremely time consuming. How can these delays be reduced? They can be reduced by making the initial set-up to serve both sinking and production and/or by sinking and equipping at the same time;

(b) Grouting and strikes: little can be said about these items;

(c) Equipping: steel installation time is a function of the accuracy required. The installation of electric cables and pipes can be greatly simplified by installing them in a location where they may be installed from a conveyance. For example, if an electric cable can be placed in a cage on a reel and played out and bolted to the shaft lining as the conveyance descends, installation will be muc faster than installing it in a special utility compartment. Pipes can be handled in a similar fashion.

The real answer to the time question in North America is to develop a method whereb sinking and equipping can be done at the same time without reducing the sinking speed or sacrificing accuracy. At present, equipping while sinking slows the sinking rate considerably and new methods will have to be devised.

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When planning a shaft, it is good to establish your permanent requirements and then before proceeding, discuss the sinking aspect with experts in that field. It may be that the sinking requirements of the headframe and shaft can be designed in a manner which will provide for both requirements at little or no extra direct cost and with a considerable ultimate savings.

Contractors do not all agree on how to sink shafts, but in the case of a deep shaft, they would agree that these points are of major importance:

(a) The bigger the shaft area and the deeper the shaft, the larger the hoist must be. Therefore, where possible, the production hoist should be made available to the contractor, at least for the final sinking stage;

(b) If the hoisting compartments are small, the possibility of equipping as you sink and maintaining a satisfactory rate of advance is eliminated as the size of the hoisting bucket is limited;

(c) In the United States, jumbo drilling in shafts is common, and while we have not used jumbos a great deal in Canada, there is no doubt that we will have to mechanize this phase of sinking. Again, if the compartment size is too small, a jumbo cannot be used and, therefore, the possibility of sinking and equipping at the same time is eliminated;

(d) In the final analysis it is always the average advance per calendar day that counts. It is less than the maximum rate but is directly correlated to the maximum rate, which in turn is determined by hoist and shaft design characteristics. The figures given here are an example of the comparison.

In closing, I would like to say that I do not set out what was done at Creighton as the only or best way to sink a deep shaft. A great deal was learned on this job, and it is hoped that by making this information available, others may benefit from the experience. In this regard, for those interested in the mechanical and electrical aspects of this project, a detailed paper was prepared, covering these items.

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VIII. UNDERGROUND MINE PLANNING AND DEVELOPMENT

R. S. Haflidson
J. Toivanen

This paper deals with various principles and factors involved in the choice of mining method, development access, production requirements and hoisting methods. To illustrate the principles, a theoretical Mine "X" is studied and planned to the production stage, using techniques and methods currently being used in Canada.

No attempt is made to cover all possible cases or provide universally applicable criteria. What works in one case does not necessarily work in another, even if the parameters appear similar, and there is danger in trying to apply successful techniques to new situations without due consideration of all the factors. However, it is quite easy to see the similarities in Canadian mining from province to province and from mine to mine. It is also quite easy to pick up similarity or recurrence of error in places where fundamentals were ignored, not thoroughly studied, or overwhelmed by economic factors.

In order to deal with our subject in one lecture, the material which follows is given in simplistic form, avoiding complexities and confining the calculations to basic We are, of course, aware that there are complexities in every mine, but these may be handled in questions form or in discussion later. It may also be noted that computer print-outs are not included in any of the calculations in this lecture for purposes of clarity. Computers are, of course, in use in many phases of mine planning in Canada and are a proven, useful tool in many instances including ore reserves, cash flows and production scheduling. In others the benefits are not as obvious and in still others their use and presence are a matter of controversy, so that at this stage their application at certain mines or for specific studies is usually a subjective decision. In any case, the calculations which follow are shown in the simplest form possible.

A. Basic parameters

In mine planning as in many other areas of endeavour, forethought wins, and the time (and money) spent in reconnaisance is seldom wasted. It can probably also be safe said that there have been more serious mistakes made in not tying down the basic parameters than there have been in developing the working concepts. The old operations-analysis maxim of "garbage in - garbage out" is most apt here, and it is also true that insufficient information can be just as disastrous or misleading as "garbage".

Part of the reconnaisance problem in Canada is that the money is not always available until a production decision has been made, and when that is done, the pressure

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is on to be turning out metal "yesterday". The end result is often too much rush and too little time for either cool study or an intelligent programme of collection of basic information. This tendency should, as far as possible, be resisted, and every effort made to gather accurate, thorough information on the basic factors which will affect the mine planning. The most important of these are as follows:

1. Financing: timing and extent. As indicated above, this factor often overwhelms all others, so it is important to know in the beginning when funds for the project are to be available and what the limits are. Where the financing is internal or to a company subsidiary, this is usually well done, with good communication between the engineering and the financial departments. However, when the money comes from stock issues, bonds or bank loans, some uncertainties are introduced which complicate or frustrate the planning to the point of "playing by ear". In such instances the engineers can contribute greatly by an early statement of what can or cannot be done and by firm advice of timing and requirements.

2. Product market and price. It is usual in Canada for the owners of a potential mine to employ specialists in the marketing of the product (or products) and there are many good ones available with world-wide connexions who can provide quick information and prospective buyers. It is usually wise to get a smelter or sales contract in hand as early as possible so that feasibility cash flows are accurate. At the same time, it is vitally necessary to make a forecast as to changes in product price in the future. These studies are complex when an effort is made to judge the effect of such factors as politics, war or labour strife. More often than not, simple extrapolations of the past or the current prices are used, and while it is important to be reasonably conservative on price predictions, there have been many good mines turned down by an over-conservative estimator.

3. Ore reserves. This aspect in mine planning is obviously critical because the rate of mining, plant size and pay-back period will all depend upon it, and while the subject has been adequately covered in a previous lecture, some comments are relevant to our subject.

There is considerable difference of opinion among Canadian mining companies and engineers regarding general theory, methods of application, the definition of ore and the classification of reserves. In practice, one observes calculations ranging from the "painfully conservative" to the "flagrantly promotional", all under the heading of ore reserves. Personally, we prefer for the purposes of planning, the definitions given by the Association of Professional Engineers for the Province of Ontario and published in their performance standards as follows:

"Ore" is a natural aggregate of one or more minerals which, at a specified time and place, may be mined and sold at a profit, or from which some part may be profitably separated.

"Proven ore" or "measured ore" is that material for which tonnage is computed from dimensions revealed in outcrops or trenches or underground workings and/or drill-holes and for which the grade is computed from the results of adequate sampling. The sites for inspections, sampling, and measurement are so spaced and the geological character so well defined that the size, shape and mineral content are established. The computed tonnage and grade are judged to be accurate within limits which must be stated. It must

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be stated whether the tonnage and grade of "proven" or "measured" ore is in situ or extractable, with dilution factors shown, and reasons for the use of these dilution factors clearly explained.

"Probable ore" or "indicated ore" is that material for which tonnage and grade ar computed partly from specific measurements, samples, or production data, and partly from projection for a reasonable distance on geologic evidence. The sites available for inspections, measurements and sampling are too widely or otherwise inappropriately spaced to outline the material completely or to establish its grade throughout.

"Possible ore" or "inferred ore" is that material for which quantitative estimate are based largely on broad knowledge of the geologic character of the deposit and for which there are few, if any, samples or measurements. The estimates are based on an assumed continuity or repetition for which there are reasonable geological indications these indications may include comparison with deposits of similar type. Bodies that are completely concealed may be included if there is specific evidence of their presen

Given acceptance of the above classifications and adherence to them, one is still faced with judgement as to what can be included in planning calculations. In general, some of the "probable" category will be included and most of the "possible" excluded. However, the engineer, if he is performing his function, should in the end be satisfie that the tonnage which he has included will with "reasonable certainty" be run through the mill. On the other hand, over-cautiousness can be unfair and unrealistic. Intuit and experience count for a lot here and it is most important that sufficient drilling and/or development be done to allow proper calculations and prudent judgements.

4. Dimensions and attitude of orebody. In addition to tonnage and grade for ore reserve purposes, there must be sufficient development and definition drilling done to tie down the length, width, thickness, dip, plunge, faulting and variances - all of which will affect the decisions regarding access, development and mining method.

5. Strength properties of the ore and wall rocks. In mines that have been in product for some time, usually sufficient experience has been gained to make good assessments. However, where new mining situations are being planned, drill core may not provide sufficient information for proper design of stope and pillar dimensions on the one han or of the caving characteristics of the wall rocks on the other. From this point of v alone, it is usually most desirable to create some openings in ore and wall rock early in the programme and carry on some preliminary testing and rock mechanics work.

6. Surface features. Any mine will sooner or later be affected by surface character and this aspect should receive early attention. Topography and overburden contour maps should be available as well as the related geological and outcrop information. Where subsidence or the effects of mining in an inhabited area are a factor, soil studies should be made to determine slope requirements and stability. Ground water should be investigated and where the existence of ground water into mine formations is indicated a grouting programme should be completed before underground openings are created.

7. Mineralogy and metallurgy. It is usual during the final stages of a successful exploration programme to run studies on the core to resolve geological terminology and minerals present. It is also usual at this time (or shortly after), to start metallurgical test work to resolve recoveries, concentrate grades and grindability. These studies also provide useful information in determining the presence of minerals which may affect the mining techniques (talc, chlorite, pyrrhotite) and in resolving the availability of tailings for backfill and the precolation rates.

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B. Choice of mining method

In approaching this phase, it is important that the personnel involved retain an open mind, unfettered by fads or personal bias. If there has been past mining experience in the orebody, or if it is an ongoing operation, there will be heavy prejudice in favour of retaining the same system for the simple reason that mine personnel are trained in the techniques and the equipment has been bought. These are worthy reasons for continuing with a method, but they should not be allowed to overwhelm good sense or prevail where there are valid reasons for a change. In such instances it is good practice to retain at least one experimental stope on the basis of trying new methods, new equipment and new techniques, in order to keep the staff flexible and provide accurate reinforcement of argument for change. In our view this is not done enough in Canadian mining.

There is another tendency at work, and that is to over-apply a method which has been markedly successful in other mines and other camps. The open blasthole method, for instance, has been used again and again where ground competence indicated a less open method or at least a tighter basis. The resulting heavy dilution and out-of-control caving can be an expensive price to pay for reduced mining cost per ton.

Speaking very generally, and provided, at least, that some of the requisite conditions are there, Canadian preference of mining method would probably take the following order:

Method	Relative mining Cost/ton	Requisite conditions
Open pit	$1.20	Ample dimensions, low overburden
Open blasthole	$3.00	Fair dimensions, competent ground
Shrinkage	$3.50	Steep dip, good walls
Room and pillar	$3.50	Flat dip, fair ground
Sublevel retreat	$4.00	Steep dip, caving qualities
Cut-and-fill	$6.00	Cheap and adequate source of fill

C. Determination of tonnage rate

It should be said in the beginning that mining rate determination is more than a calculation and the final fix will have to stand up to the practical problems in mining the orebody. Thus, once the optimum range is established, studies should be initiated on scheduling drill and blast cycles and the establishment of the possible all along the line. In vertical orebodies the old rule of thumb used to be one half to two thirds of the tons per vertical foot. In flat-lying orebodies this rule of thumb was modified to the tonnage which a one-foot slice along the strike of the orebody would give. These were the roughest approximations but they worked out quite well in many cases.

There are also other factors beyond a straight mechanical determination of the optimum monetary rate to exploit the orebody, and these usually involve the "corporate

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social responsibility" of the mining company. Effects on the ecology, the duration of personnel employment and the impact on the existing culture are all matters of importa which might override whatever answer is arrived at by economic calculation. However, lecture is concerned primarily with the technological aspects and the considerations which follow are therefore profit-oriented.

It will be noted from the calculations on cash flow (see annex), that there are two major principles affecting the outcome:

(a) The larger the tonnage rate, the greater the plant expenditure and interest charges;

(b) The larger the tonnage rate, the smaller the operating cost and thus the greater the operating profit.

The example shown in figure 8 illustrates the usefulness of plotting rate of return against tonnage rather than present value. In this instance the rate of return is optimum from 1,000 to 1,200 tons per day but change of direction is not as marked. The weight of each of these factors will vary depending on the many variables in each case.

D. Choice of access and hoisting method

Decisions regarding choice of access will depend initially on whether the prime purpose is one of development or production, or both. The obvious choices in Canadian practice are (a) by adit (flat or declined); and (b) by shaft (vertical or inclined). With the advent of load-haul-dump techniques and the evolution of related equipment, the declined adit (ramp) has become very popular both for purposes of developing the orebody and for service access during production later. In large-tonnage situations it is quite common to see both, combining the advantages of fast developing and vehicle service with low-cost hoisting. Several smaller operations have no shaft and use ramp haulage for hoisting, but these usually evolve into shaft hoisting at depths below 500 feet.

There are also combinations of ramp plus conveyor, in some instances with the ramp and conveyor in separate openings and others sharing the same heading. These are not as numerous as the ramp plus shaft combination and are usually special situations.

In order to compare the advantages and disadvantages involved in each mode of entr it is useful to make calculations showing the capital, operating and servicing costs fo each method. Such a calculation is shown in the annex.

For this exercise it is assumed that the project will be financed by borrowing at 8 per cent, and the loan taken down over a three-year period and then amortized over a four-year period. In order to provide a consistent comparison this was done in each of the cases shown (see table 6).

E. Example: development of mine "X"

Parameters

This is a flat-lying copper deposit discovered by airborne geophysical methods and confirmed by ground geophysics followed by detailed diamond drilling. Strike length

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is 500 feet, thickness 20 feet and dip 30 degrees. Proven and probable reserves indicate 1,500,000 tons "reasonably assured" to 1,000 feet vertically grading, 2.5 per cent copper with some gold and silver. The orebody is open to depth.

If the deposit can be brought to the production stage financing will be by bank loan at 8 per cent.

Market studies confirm the existence of reliable buyers of concentrate grading 25 per cent copper or better, with post-concentrator costs indicated in the order of 11 cents per pound of copper. Studies on copper price during the assured life of the mine indicate a low of 45 cents, a high of 75 cents and a probable average in the area of 57 cents.

The ore is massive sulphides, mostly pyrite, quite competent with an excellent andesite hanging wall and a rhyolite footwall. The zone is cut by minor faults and diorite dikes with some ore displacement and structural weakness at the contacts.

Surface topography is flat with low overburden (20 feet) and no major water courses. Access by road is easy and there are suitable areas for a mining plant. Early drilling indicated the presence of ground water under pressure so a grouting programme will be required before underground development commences.

Bench tests on sections of drill core indicate reasonably good metallurgy with copper recovery at 94 per cent in a 25 per cent concentrate.

Feasibility

Cash flow projections are then run using the above parameters and rough approximations on costs. Accurate costs cannot be estimated until tonnage rate and mining method are determined. From these approximations a viable operation with a pay-back period of four years and an optimum milling rate of 1,000 tons per day is indicated.

Mining method and costs

The flat dip of the orebody along with the narrow thickness and competent hanging wall point to a panel mining method (inclined room and pillar) with 50-foot wide topes and 20-foot pillars to start. Drilling in the stopes will be with air legs and the ore slushed to the level to be picked up by LHD equipment. On this basis, mining costs are estimated approximately as in table 4.

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Table 4. Mining costs

Item	Development	Mining and stope preparation	Total
Labour			
Drill and blast	.19	.45	.64
Mucking and hoisting	.12	.40	.52
Pipe and rockbolting	.04	.04	.08
Scaling	.03	.03	.06
Miscellaneous	.02	.03	.05
Subtotal	$ .40	$ .95	$1.35
Materials and services			
Explosives	.18	.20	.38
Air and water	.06	.21	.27
LHD fuel and parts	.15	.36	.51
Rock drills	.01	.09	.10
Drill steel	.04	.06	.10
Slushers	.07	.14	.21
Shops and services	.52	1.37	1.89
Subtotal	1.03	2.43	3.46
Grand total	$1.43	$3.38	$4.81

Access development: mine "X"

For the development concept, two purposes had to be kept in mind. First, provin up the orebody by headings in ore and second, efficient mine operation on a 1,000-ton-per-day production basis. From the calculations (see section F), the shaft concept ha the overwhelming advantage on capital requirements and operating efficiency below 500 feet depth. On the other hand, the ramp alternative would give quick initial development of the orebody and result in production from the upper levels much earlie thus bringing a production date ahead by at least six months.

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A third alternative provides the advantages of both the ramp and the shaft concept. This incorporates the use of the ramp for initial development and then production to the 500-foot horizon followed by later shaft sinking. This allows time for a thorough drilling programme to establish the depth limits of the orebody, allowing proper location of a shaft and also time for a sinking programme to be completed while production is being maintained.

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Appendix

COST CALCULATIONS

A. Access development

1. Shaft sinking

Contractor's plant and preparation cost

Site preparation	$24,000
Transportation of equipment	8,000
Sinking head frame with bin	70,000
Sinking hoist 6 ft. diameter	60,000
Compressor plant	30,000
Diesel or power plant	35,000
Boilers for heating	15,000

Temporary buildings

Hoist house	$20,000	
Compressor house and power	20,000	
Dry, office-warehouse	10,000	
Garage and shops	12,000	
Powder and cap houses	5,000	
Pump house	8,000	75,000
Plant electrics (surface)		20,000
Air and water lines on surface		10,000
Miscellaneous equipment for sinking (fans, doors, small tools)		40,000
Rock work and concrete for collar		50,000
Miscellaneous labour 10 per cent of above		43,700
Contingencies and contractors fees (20%)		96,140
Total		$576,840

Sinking cost and excavation

Rock work	$200/ft	
Timber	60/ft	
Rock bolts and pipe	30/ft	
Landings and ladder	10/ft	
Total	$300/ft x 1,300 ft	390,000
Add stations 6 x 8,500 =	$51,000	
LP and spill pocket =	50,000	101,000
Total contract cost of shaft		$1,067,840

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Appendix (continued)

Surface plant

Headframe and shaft equipment

Excavation and foundations	$19,000		
Steel	69,000		
Ore bin steel	18,500		
Waste bin steel	9,500		
			$116,000

Erection

Steel (structural)	$43,000		
Deck house	27,500		
Ore bin	9,000		
Waste bin	6,000		
			$85,500

Sheathing			$14,800
Pipes, wiring, services			9,000
Sheave wheels			4,000
Skip-cage units (3)			24,000
Ropes 2 x 1,700' @ $2.00			6,800
		Total	$260,100

Hoist room

Excavations and foundations	$22,000		
Building	27,000		
			$49,000

Equipment

Hoist foundations	$ 7,750		
10' diameter hoist (used)	175,000		
Installation	12,000		
Electrics	5,250		
Compressor and piping	30,000		
			$230,000
		Total	$279,000

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Appendix (continued)

Power supply

Electric cables	$18,500	
Substation electrics	2,500	
Cable trenches	9,000	
Installation	7,000	
		$37,000
	Total surface plant	576,100
	Shaft contract	1,067,840
	Total project	$1,643,940

2. Conveyor decline

Contractor's plant and preparation costs

Site preparation		$24,000
Transportation of equipment		8,000
Compressor plant		30,000
Power plant or diesel		20,000
Boiler for heating		15,000

Temporary buildings

Compressor and power	$18,000	
Dry, office-warehouse	10,000	
Garage, shop and service	25,000	
Powder and caps	5,000	
Pump house	8,000	
Portal cover (temporary)	15,000	
		$81,000
Plant electrics (surface)		$ 9,000
Air and water lines (surface)		10,000
Miscellaneous equipment and small tools		20,000
Excavation and concrete work (portal)		30,000
Miscellaneous labour 10% of above		24,700
Contingencies and contractors fees (20%)		54,340
	Total	$326,040

Decline drive and excavations

Average cost per foot rock work $200/ft 4,445 ft. @ $200		$889,000
Loading chute excavation		10,000
Transfer point excavation after 500' vertical depth		15,000
	Total	$914,000
	Total contract cost of decline	$1,240,040

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Appendix (continued)

Equipment

Underground conveyor

Fixed cost

Head gear and drive	$45,000		
Tail assembly	12,000		
Chute and feeder	10,000		
Transfer point assembly	25,000)	2nd Section	
Head gear and drive	45,000)		
Tail assembly	12,000)		
Electric starter and motor	17,000		
Electric starter and motor	17,000)	2nd unit	
Service vehicle	8,000		
Contingencies (20%)	36,600		
			$227,600

Cost related to length of conveyor

Frame and hangers	$15/ft		
Belt	32/ft		
Installation cost	30/ft		
Cables, lights, sprinklers	8/ft		
Contingencies (20%)	17/ft		
Total	$102/ft x 4,445'		$453,390

Surface portion of conveyor 400 ft (approx.)

Frame and deck 400 x 15	$6,000		
Head gear (included in U.G.)			
Belt 400' @ 32/ft	12,800		
Electrics (lights, signals only)	3,000		
Installation	8,000		
Contingencies (20%)	5,960		$35,760
		Equipment total	$716,750

Permanent surface plant

Ore bin

Foundations	$7,000		
Steel	18,000		
Erection	9,000		
Sheathing	5,500		
Miscellaneous	4,000		$44,000

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Appendix (continued)

Surface conveyor gallery		
Footings	$12,000	
Steel	20,000	
Erection	10,000	
Sheathing	8,000	
Contingencies (20%)	10,000	$60,000
Portal building		$38,000
Compressor plant		
Equipment	$40,000	
Building	35,000	
Contingencies	15,000	$90,000
Power supply		$30,000

Permanent buildings	$262,000
Equipment	716,750
Total plant	$978,750
Decline drive	1,240,040
Total project	$2,218,790

3. Decline for truck haulage

Contractor's plant and preparation costs		
Site preparation		$24,000
Transportation of equipment		8,000
Compressor plant		30,000
Power plant		20,000
Boiler for heating		15,000
Temporary buildings		
Compressor and power	$18,000	
Dry, office-warehouse	10,000	
Garage, shops and service	25,000	
Powder and caps	5,000	
Pump house	8,000	
Portal cover	15,000	$81,000

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Appendix (continued)

Temporary buildings (continued)

Plant electrics (surface)		$9,000
Air and water lines (surface)		10,000
Miscellaneous equipment and small tools		20,000
Excavation and concrete for portal		30,000
Miscellaneous labour (10%)		24,700
Contingencies and contractors' fees (20%)		54,340
	Total	$326,040

Decline drive and excavations

Average cost for rock work $200/ft 5,860 ft @ $200		$1,172,000
Slashing for passing points 200 ft x $200		40,000
		1,212,000
Loading chute excavation		10,000
Surface conveyor feed chute excavation		15,000
	Total	$1,237,000
	Total contract cost of decline	$1,563,040

Surface plant

Ore bin

Foundations	$7,000	
Steel	18,500	
Erection	9,000	
Sheathing	5,500	
Miscellaneous	4,000	$44,000

Surface conveyor gallery

Footings	$12,000	
Steel	20,000	
Erection	10,000	
Sheathing	8,000	
Contingencies	10,000	$60,000

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Appendix (continued)

Surface conveyor, 400 ft (approx.)

Frame and deck	$6,000	
Head gear and drive	20,000	
Tail assembly	6,000	
Belt, 400 ft @ $28.00	11,200	
Electrics	7,500	
Installation	13,500	
Contingencies	12,800	$77,000
Portal building		$38,000
Truck dump and feed chute		25,000
Compressor plant		90,000
Power supply		20,000
Total plant		$354,000

Production equipment

Trucks for ore haulage, 4 @ $65,000	$260,000
Service garage	50,000
	$310,000
Total plant and equipment	664,000
Decline drive and excavations	1,563,040
Total project	$2,227,040

B. Operating costs: production and mine service

1. By hoisting

Labour (monthly)

	Service	Production	Total
Operating labour	$3,600	$3,150	$6,750
Maintenance labour	900	700	1,600
Total labour	$4,500	$3,850	$8,350

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Appendix (continued)

Supplies (monthly)

	Service	Production	Total
Cable	$ 170	$ 330	$ 500
Electric	50	100	150
Other	90	125	215
Power	137	390	527
Total supplies	$ 447	$ 945	$1,392
Total operating	$ 4,947	$4,795	$9,742
Cost/ton (30,000)	0.165	0.160	0.325
Service to workplaces (estimated)	0.200		0.200
Total	0.365	0.160	0.525

2. By conveyor

Operating labour (from statistics), service (minor repairs) 0.06/ton

Mine service

Labour

2 men per shift @ 2 shifts @ $36	$144/day		
Service vehicle operating cost, $4/hr	96/day		
	$240/day or $5,200/month		0.173/ton
		Total	0.233/ton

3. By ramp haulage

Hauling cost from 1,150 ft vertical distance below surface

Need 3 units @ 160 tons/unit = 480 tons/shift
At 3 shift operation per day: 3 x 480 = 1,440 tons/day

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Appendix (continued)

Labour cost: 3 x $34 = $102/shift

Truck operating cost @ $10/operating hour: 3 x 6.5 x $10 = $195/shift

Total cost = $297/shift

Cost per ton hauled = $\frac{\$297 \times 3}{1,400}$ = 0.636

Mine service cost

Same as for conveyor decline 0.173

Total cost $0.809

C. Secondary development requirements for comparative purposes

Shaft

Level drives to ore: 2,450 ft @ $100 = $245,000

Conveyor decline

Level drives: 700 ft @ $100 = $ 70,000

Ramp haulage

Level drives: 1,100 ft @ $100 = $111,000

D. Summary

Established preproduction costs for mining to 1,000' elevation	Method of access		
	Shaft	Conveyor ramp	Truck haulage ramp
Permanent plant and equipment	$ 576,000	$ 978,750	$ 664,000
Primary development	1,067,840	1,240,040	1,563,040
Secondary development	245,000	70,000	111,000
Totals	$1,888,940	$2,288,790	$2,338,040

E. Preproduction costs estimated for various mining depths

In the following tables the previously established costs for mining down to 1,000' vertical depth were used as a standard. For shallower depths allowances were made to the fixed and variable costs as influenced by depth.

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Appendix (continued)

Table 5. Costs related to mode of entry

Method of access	Vertical mining depth (feet)	Plant, equipment and primary development	Secondary development	Total cost
		(dollars)		
By vertical shaft				
	400	950 000	100 000	1 050 000
	600	1 200 000	145 000	1 345 000
	800	1 432 000	180 000	1 612 000
	1 000	1 644 000	245 000	1 889 000
By conveyor decline				
	400	1 263 000	40 000	1 303 000
	600	1 590 000	50 000	1 640 000
	800	1 946 000	60 000	2 006 000
	1 000	2 219 000	70 000	2 289 000
By truck haulage ramp				
	400[a]	1 040 000		1 040 000
	600	1 322 000	10 000	1 332 000
	800	1 948 000	30 000	1 978 000
	1 000	2 227 000	110 000	2 337 000

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[a] For these depths ore is hauled directly from the stoping areas to a surface crusher. Primary development costs can be substantially reduced by eliminating the development work below last mining level and the ore pass.

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Appendix (continued)

Table 6. Preproduction and capital costs per ton amortized over four years

(Mining one thousand tons per day)

Mining depth	Total loan over 3 years	Accumulated interest charges	Total capital	Annual mortgage payment, 4 years @ 8 per cent	Cost per ton
	(dollars)				
			Vertical shaft		
400'	1 050 000	145 990	1 195 990	361 093	1.00
600'	1 345 000	189 365	1 534 365	463 255	1.29
800'	1 612 000	226 950	1 838 950	555 220	1.54
1000'	1 889 000	265 955	2 154 955	650 625	1.81
			Conveyor decline		
400'	1 303 000	183 450	1 486 450	448 790	1.25
600'	1 640 000	230 895	1 870 895	564 860	1.57
800'	2 006 000	282 425	2 288 425	690 920	1.92
1000'	2 289 000	322 270	2 611 270	788 395	2.19
			Truck haulage ramp		
400'	1 040 000	146 425	1 186 425	358 205	1.00
600'	1 332 000	187 530	1 519 530	458 775	1.27
800'	1 978 000	278 485	2 256 485	681 280	1.89
1000'	2 337 000	329 030	2 666 030	804 925	2.24

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Appendix (continued)

Table 7. Summary of preproduction and operating costs related to modes of entry

(dollars)

	Vertical shaft			Conveyor decline			Truck haulage ramp		
Mining depth	Mortgage cost	Operating and services	Total cost	Mortgage cost	Operating and services	Total cost	Mortgage cost	Operating and services	Total cost
400'	1.00	0.48	1.48	1.25	0.20	1.45	1.00	0.40	1.40
600'	1.29	0.49	1.78	1.57	0.22	1.79	1.27	0.55	1.82
800'	1.54	0.50	2.04	1.92	0.22	2.14	1.89	0.70	2.59
1000'	1.81	0.53	2.34	2.19	0.23	2.42	2.24	0.81	3.05

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Appendix (continued)

Table 8. Summary of operating results from mining a hypothetical 3 million ton ore body at various mining rates

(In dollars)

Rate of mining	500	750	1,000	1,500	2,000
	(tons per day)				
Ore reserves	$3 000 000	$3 000 000	$3 000 000	$3 000 000	$3 000 000
Net profit per ton	$6.40	$6.90	$7.40	$8.40	$9.40
Capital and preproduction expenses	$5 000 000	$6 200 000	$7 500 000	$9 600 000	$12 000 000
Life of mine	17 yrs.	12 yrs.	9 yrs.	6 yrs.	5 yrs.
Payback of capital	6th yr.	5th yr.	4th yr.	3rd yr.	2nd yr.
	(In thousands)				
Total operating profit	$19 200	$20 700	$22 200	$25 200	$28 200
Less interest exp. @ 8%	1 395	1 294	1 355	1 375	1 455
Net earnings	17 805	19 406	20 845	23 825	26 745
Debt retired	5 000	6 200	7 500	9 600	12 000
Available for PV calculations	12 805	13 206	12 745	14 225	14 745
PV discount rate which equals preprod expenses	8.75%	10.1%	9.7%	9.2%	5.9%

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IX. TRANSPORT OF MINERALS

F. H. Skelding*

The traditional attitude of the mineral producer toward the handling and transport of minerals outside the mine-mill complex is easily expressed. Responsibility for transportation of mineral products begins at the mill and ends at the nearest transfer point at which the material can be handed over to a carrier such as a railroad or a steamship line. In other words, when designing a new project, the typical miner will install a road, a conveyor, a ropeway or a spur railway system from the mine to the nearest rail or water terminal. Rarely if ever have the characteristics of the market for mineral products been taken into account and the subject of mineral transportation considered as an integrated whole. It is time this kind of thinking changed.

The very fact of this seminar is clear proof of the considerable technologic advances that have been made in all aspects of mineral exploration, development, production and processing. These together with the really remarkable recent developments in the technology of handling and transporting minerals have had a significant effect upon the mineral industries of the world and have created the conditions that make it increasingly necessary for the producer to become much more involved in matters of mineral transportation than ever before. In this paper the subject of mineral transportation technology is discussed in terms of the significant technical and economic characteristics of the various modes, and the major variables that govern the applications of transportation technology to actual projects.

Much of what is said may appear to be obvious, but even a cursory examination of mineral transportation in today's world mining industry shows clearly that the obvious is usually overlooked. That this is so is all the more interesting, because for many commodities, the cost of transportation comprises as much as 70-80 per cent of the total cost of the material delivered to the consumer. The notable exceptions are the high unit value commodities such as copper concentrates. Direct transportation charges are not likely to be more than 3-8 per cent of the cost of concentrates worth $250-$300 per short ton. Indirect costs, however, such as wind losses from open railroad cars in transit (2-3 per cent of the carload by weight), spillage, contamination, and other disadvantages of the more traditional methods of transport, can add up to a significant total.

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* Director, Corporate Planning, Fluor Utah, Inc.

In this discussion, the significant technical and economic features of each of the modes of transport of most concern are briefly reviewed and important recent technologic advances noted. The various modes are then subjected to a comparative analysis in terms of the variables governing their application and the selection criteria commonly employed. The primary objective of this paper is to discuss the applications of modern transportation technology to the mineral industries as an integrated system. This concept itself, integrated transport systems, is the single most important development in mineral transportation in the last 20 years, and it is having a profound effect upon the industry.

A. Overland transportation systems

This part of the discussion begins with and gives most attention to the various forms of overland transport. In many cases, these represent more or less viable alternatives whereas waterborne and overland systems are not basically competitors. A railroad usually cannot be substituted for a steamship. The obvious exception, of course, is the circumstance in which transport by inland waterway or coastal shipping is an alternative to overland movement. Other modes are also considered, however, because it is increasingly common today for mineral shipments to move very long distances which almost certainly entails utilization of several different modes. Consequently, a broad knowledge of mineral transportation generally enables the mineral producer to design those parts of the whole transport system for which he is responsible in a way to achieve greatest compatibility with the other components, thereby maximizing efficiency and minimizing costs. All this again highlights the value of considering mineral transport as an integrated whole.

Road transport

Over-all, road transport is perhaps the commonest and at the same time the least known form of mineral transport. Most producers have never carefully analysed their vehicular operations, although this is beginning to change. For the present, however, it means that relatively little useful data is available. Vehicles used for carrying mineral products are of many types and capacities, the latter ranging from 10 to 45-50 tons. The vehicles may be single units with or without trailers or they may be tractor-trailor combinations or a mixture comprising up to three units. They may be fixed bed or dumping units discharging from side, end or bottom.

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There have been no significant technologic advances in road transport recently, but there has been a great deal of improvement on existing technology, especially in the area of bulk transportation. Advances in design and evolution of new alloys for lightweight vehicles have improved the net-to-tare ratio while simultaneously reducing maintenance and repair costs. Increasingly powerful, lightweight engines have been developed. The diesel has largely replaced the gasoline engine. The gas turbine is approaching commercial utility (that will be a real advance) and ways of using engines in tandem have been developed. Gear trains have been improved although automatic shifts are not particularly useful in bulk transport over mediocre roads. Cabs and controls are being designed to reduce driver fatigue but enhance visibility and safety. Continual improvements are being made in tires which decrease their susceptibility to damage and increase their cooling ability. Inasmuch as tire costs can account for as much as one third of truck operating costs, this is no small achievement. All wheel drives are coming into use and, as in the case of the Kitwe (Zambia)-Dar es Salaam (Tanzania) copper run, are proving very practicable.

A special area that borders on technologic breakthrough is the development of the pressure differential (PD), "pneumatic" or "air-glide" vehicles. These are specially designed tank trucks to carry very finely divided dry materials such as talc, cement, clay or gypsum which are usually very difficult to handle due to their tendency to pack rather than to run freely out of a bin or chute. In the PD system, the bottom unloading steel or aluminium tank is slightly pressurized to about 15 pounds per square inch (psi) by blowing the air up through the mineral to loosen it. Pressure in the discharge pipe is kept 2-3 psi below that in the tank, causing the mineral, already loosened by the entrained air, to flow or slide. The method prevents contamination and, by keeping the mineral thoroughly mixed, prevents the grain size segregation which is a natural tendency of this type of material under gravity flow.

Roads are not usually thought of as being an integral part of a road haul system, but clearly they are. Surface and gradient are the important factors. Road surface conditions determine the gross weight hence payload that can be carried as well as the number of trips that can be made between terminals. The problem is that the total weight of the vehicle and load is in contact with the bearing surface through the wheels which means high point loadings. To some extent, poor surface can be compensated by wheel and axle configuration, but the possibilities are limited. Surface also directly affects speed, irrespective of load. It also has a significant effect on maintenance costs and tire wear. A dirt road, for instance, can reduce tire life and increase maintenance cost by 50 per cent.

Gradient and curvature are also extremely important. Road vehicles can, according to type, negotiate surprisingly steep grades, but 6-8 per cent is the normal limit, with short stretches of 10 per cent, the limit for most practical purposes. The limit of curvature varies according to vehicle type,

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but a 50-ft radius would certainly be the minimum for any but a small vehicle and well beyond the capability of most.

A number of very innovative, cross-country or off-road vehicles have been developed, including the rollegon which, instead of having tires or tracks, travels on very large, low-pressure inflated bags, and the air cushion vehicles which are lifted and propelled on a column of air and are thereby totally independent of the surface. None of these methods has as yet proved to be of any economic value for the movement of ores or concentrates although they are invaluable for exploration and for the movement of light cargo and personnel.

Road transport costs are not low, and unlike those for the rail, maritime or pipeline methods, the unit cost (cost per ton mile) of road transport for a given tonnage first declines with distance, then begins to climb again giving a U-shaped curve like that for air transport. Typical costs are 4.5-5.5 cents per ton mile.

Belt conveyors, aerial ropeways, and related methods

Over the last few years, improvements in the technology of belt conveyors have led to the use of this method, once limited to in-plant materials handling, for moderately long distance hauls of up to 60-70 miles. Conversely, the aerial ropeway or tramway, one of the oldest transport systems, has not undergone any technologic revolutions but is finding increasing application in many parts of the world. The real technologic advances in this area have come in the development of new systems such as Dashaveyor, Seccam and Serpentix which are attempts to overcome inherent deficiencies in the belt conveyor and, to some extent, the ropeway.

Belt conveyors. The classic, ubiquitous, multi-ply belt conveyor is not employed for long distance haulage. The limiting factor is tensile strength, commonly measured in pounds per inch of belt width. The great weight of a conventional belt relative to its tensile strength, together with the massive support structure required to carry it and the limited flexibility of this type of belt which severely limits the vertical and horizontal corners it can turn, restrict its use to in-plant, short distance operations.

The wire cored belt, a fairly recent innovation, allows much longer spans or flights and is the development which led to the use of conveyors for moderate to long distance haulage. Such a belt can be given a tensile strength of 3,000 pounds or more per inch of belt width, in contrast to the 50-pound tensile strength for belts of 20 years ago.

Most important however has been the very recent development of the so-called cable belt system. In this method the conveyor belt, transversely strengthened with steel rods, rides on top of two parallel cables. It is the cables, not the

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belt, which are tensioned and it is the cables, not the belt, which are driven. Span or flights of up to 10 miles, perhaps more, are possible. The cables ride in sheave wheels and there are no troughing or return idlers and only about one-quarter the number of bearings or rolling surfaces found on conventional conveyors.

Conveyor belt capacity is primarily a function of belt width and speed. Speed in turn is dependent upon particle size and size distribution, density, and moisture content. Belt capacity is also affected by the troughing angle. Merely increasing the angle by 10°-15° from 20° increases capacity by 10-25 per cent. A trough angle of 45°, however, is no help due to excessive flexing and wear on the belt.

Aerial ropeways (tramways). Aerial ropeways, also known as aerial tramways, are among the older methods of mineral transport. They are of two types. In the monocable system, the single endless cable or rope serves as both track (support for the load) and motive power. In the bicable system, two, often four cables in parallel serve as the supporting or track ropes upon which the carriage of the load carrying vehicle moves under motive power supplied by an independent (third) rope. Either type may be reversible, but if they are, the system is only one span long, whereas a one-way system may be of considerable length comprising several spans.

The bicable can carry a greater load (to five tons per unit) at greater speeds, hence can reach capacities of $\geq$ 500 tph although 250-350 tph is more usual. The monocable carries a lighter load (one ton) at about 600 feet per minute for typical capacities of 200-300 tph. A bicable can climb a steeper gradient and use longer unsupported spans (up to a mile) than the monocable.

Capital costs for aerial ropeways are somewhat less than those for belt conveyors although of the same order of magnitude. Monocables, as might be expected, are less costly (30-40 per cent less) than bicables. Unit operating costs of ropeways, however, are about twice those of a conveyor or more. For example, the 0.5 million metric ton per year capacity, 60 mile Swedish base metal concentrates ropeway requires an operating crew of 21. The 3 million tpy Jamaica 6.5 mile bauxite belt conveyor requires a crew of 20.

Other systems. Several new systems, more or less related to the conveyor, have been devised in an effort to overcome the weaknesses of conveyors and ropeways. The three best known are probably Dashaveyor, Seccam and Serpintex.

The Dashaveyor system basically consists of self-powered modules or cars, operated in small trains, or flights of 10-50 modules running at high speed (40 mph) on light tracks inside a box section that prevents derailing designed to turn through 360° tight radius vertical or horizontal circles. It has not proved very successful.

Seccam is quite similar to Dashaveyor in that it consists of small cars or modules travelling on rails in trains or flights of 200 or more units. Motive power is provided by in-place stations along the route. Several installations are in service in New Caledonia.

The Serpentix system was developed in Germany and in essence consists of a flexible, pan conveyor system capable of turning very sharp vertical or horizontal corners and making what amount to vertical lifts by climbing a tight spiral. It is not, so far as known, used for long distance hauling.

Railroad transport

The two basic transport systems just discussed are essentially for short to medium-length hauls (5-50 miles) of relatively small to medium volumes, typically less than 1 million tons per year. Railraods are particularly known for long distance hauls of any but preferably large quantities of material.

There have been no significant technologic advances of "breakthrough" quality in rail haulage over the past decade or so. Such advances that have been made are very similar in kind to those already described under road transport; namely improved materials and designs to achieve more powerful but lighter engines, improved net-to-tare ratios for cars (that is, more payload within maximum permissible gross weight), and the like. The most significant development in railroad transport in many years, the one that has literally revolutionized the system in North America and to a lesser extent (at least potentially) in any country with an extensive existing railroad system, is a development not in technology, but in management. This was the introduction of the unit or integral train system.

In brief, one or more trains, consisting of about 100 cars of about one hundred ton capacity, permanently coupled together and assigned more or less permanent prime mover units, travel as nearly as possible continuously between two terminals, loading at one, discharging at the other. Given a sufficient annual ton-mile volume, the cost of unit train haulage can be as low as 0.25 cents per ton-mile. There are, of course, many variations on the basic concept just outlined, but the principal remains the same - maximum utilization of the railroad plant in revenue service. Rolling stock is often specially designed not only for this type of service but for a specific haul. Emphasis tends to be upon rapid load and discharge in order to minimize turn-around time.

The unit train concept as outlined is fundamentally the same as the case of a long haul rail system designed and built specifically to serve a new, large mineral development like those in western Australia. The basic difference is that one is strictly project-oriented which means it must bear the full capital costs of roadbed, rolling stock and ancillary items. The other, the

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unit train as first described, is simply a way of maximizing utilization of the existing equipment and roadbed of a general service rail system.

The critical variables governing railroad haulage are grade and curvature. The advantage of rail over road is that on level track it requires but 4 pounds of force to move one ton of payload compared to the 20 pounds of force needed to move one ton on the highway. If, however, the railroad has a positive grade of only 2 per cent, the force required to move one ton is increased tenfold whereas under identical highway conditions, the increase is only treble or quadruple. Railroads endeavour to limit grades to 2 per cent and never to exceed 3 per cent although there are of course occasional exceptions.

Curvature too is significant. Curve resistance is 0.8 lb per ton per degree of curve. This would come to four tons per degree of curvature for a 10,000 ton train. Not only is efficiency seriously affected by unduly sharp curves and excessive curvature along a stretch of track, but maintenance costs both for the rails and the rolling stock, primarily wheels, increase exponentially. Railroads attempt not to exceed 10° of curvature.

Rolling stock used for mineral haulage varies. Probably most common and convenient for bulk mineral transport are the covered or open gondola (solid bottom) and hopper (bottom dumping) cars. The box car, and to a more limited extent, the flat car are used for shipment of minerals in bags or drums, pallets or containers. PD (pressure differential) cars are becoming quite common. They epitomize one advantage of rail transport. The cars can, for a short period of time, be used as a fairly inexpensive form of storage. PD cement cars, for instance, can be run onto any convenient siding to serve as a local distribution terminal to several large construction projects in the area. PD highway vehicles provide the shuttle from rail to jobsite. Diesel-electric and diesel-hydraulic prime movers are most common. Electric motive power is not usual on long hauls and the few remaining steam locomotives are rapidly being phased out.

Capital costs for mainline railroad track typically run $500,000 per mile and up. In addition, many ancillary facilities are required such as terminal and storage space, switching yards, maintenance shops, etc. Rolling stock is no small item. A typical line haul (3,000 hp) diesel engine costs $250,000 or more and cars (trucks) vary anywhere from $16,000 for a basic gondola or box car to $30,000 for a side dump. A tank car can cost from $20,000 to $200,000; the latter being a pressurized, heated or refrigerated, lined, corrosion resistant special design vehicle.

Operating costs for railroads tend to be somewhat high. More or less 50 per cent of total operating costs, including capital amortization, represent wages and of the remainder, a large proportion is attributable to predominantly consumption items of various kinds such as fuel and wear items.

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Slurry pipelines

Probably the most important single technologic development of the last several decades in terms of impact upon mineral development is the slurry pipeline. The concept is not new, having been employed in the California goldfields and in England for coal before the turn of the century. Only since 1955, however, has the long distance slurry pipeline become a commercial reality. Today, throughout the world, there are close to or possibly even more than 100 long distance (five miles or more) slurry pipelines in service, under construction, in advance planning or under very serious consideration for new mining projects.

The advantages, disadvantages and selection criteria for slurry pipelines will be discussed later. Only the major technical characteristics will be described here.

The only kind of solid slurry line presently in use is the continuous flow type carrying a single, finely divided commodity (ore or concentrate) in a liquid medium. Water is most commonly used, but plans are afoot to employ petroleum or other valuable commodities as the carrier as a means of further maximizing the economic benefits to be obtained from pipelines. Experimentation is under way with capsule pipelines. The object is to encapsulate commodities either because they are soluble or corrosive, or as a means of shipping different commodities through the same line. No commercial capsule lines are known. Finally, experiments are also being made with batch pipelining which is a simple adaption to slurry pipelines of a technique already common in the petroleum industry. In the batch line, slurries of different commodities, possibly even in different media, are separated by inert slugs of material that interacts with the components of neither of the batches it separates.

Technically, any particulate matter is a potential candidate for slurry pipeline transport, providing a suitable carrier can be found. In fact, there are many limitations. For example, the mineral may be required in grain size, coarser than can be pipelined.

Pipeline capacity depends primarily upon the internal diameter of the pipe and velocity of the slurry. The latter is usually the limiting factor because at too low a velocity (typically below 4 feet per second) the solids will settle out and plug the line, and at too high a velocity (typically anything much over 7 feet per second) internal abrasion starts to become a serious problem. Clearly, velocity is in turn a function of particle size and, equally important, the range and distribution of sizes. A homogeneous slurry having particles almost uniform in size is one thing. A slurry with a wide range of

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particle sizes is something else again. The critical velocity (that at which sedimentation begins) is apparently directly related to the square root of pipe diameter. If the diameter is increased from 4 to 16 inches, for example, the critical velocity for a given slurry is doubled. However, throughput volume determines pipeline diameter. A six-inch diameter pipe will transport about one-half million tons per year but a 10 million ton throughput will require a 26-inch pipe. Velocity also plays an important role in determining costs. The pump load increases with the square of the velocity, thereby increasing operating (pumping) costs and very possibly requiring additional pumping stations in order to maintain the desired head.

Capital costs for slurry pipelines vary as much as those for other modes, being affected by such factors as route acquisition costs, site engineering and preparation. For purely first approximation purposes, the evidence to date indicates a typical long distance ore or concentrate slurry line (not tailings) would cost on the order of $120,000 per mile. Pipe cost, commonly 70-75 per cent of capital cost, varies directly with pipe diameter. Thus a slightly more refined rule of thumb estimate would be a capital cost of $3,800-$4,200 per nominal inch of pipe diameter per mile. It might be noted that pipeline installation techniques, equipment and costs are all well known since there is no practical difference between installing a slurry, a petroleum or other type of pipeline.

Operating costs are most affected by the fixed capital charges which typically account for 70-75 per cent of total transportation costs: another way, incidentally, of saying that pipeline transport is particularly insensitive to labour costs. The most important variable is probably pumping cost which can readily account for 20-30 per cent of operating costs. When these are converted to unit cost of transportation, the advantage of slurry pipelines becomes apparent. The typical slurry pipeline cost of transport is 0.4-0.6 cents per ton-mile.

The Marconaflo system

Strictly speaking, the recently developed Marconaflo system is not a method of transportation as the term is used in this discussion. It is really a materials handling technique. However, it is included here because it is clearly high among the most important technologic developments in the field of mineral transportation to have been made in the last 30 years or more, particularly in relation to mineral development in the developing nations. Above all else, it has done much to overcome a major drawback of slurry pipelines, the fact that until Marconaflo, pipelines were incompatible with other modes of transport. Now, with the advent of Marconaflo, mineral slurries can readily be handled in rail-cars, barges and ships.

In essence, the Marconaflo system is a simple means for slurrying, handling, dewatering, reslurrying and rehandling mineral slurries. The heart of the system is the marconajet, a high pressure, rotatable, water nozzle that can be

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raised or retracted from its recessed launder well in the bottom of a tank, tank car, ship's hold or other "container" into which a mineral slurry has been dumped. Sooner or later, the solids settle out of any slurry to pack into a solid mass at the bottom of whatever contains them. The Marconajet system rapidly reslurries the material so that it can be pumped on its way, a far cheaper process than digging it out. Moreover, the Marconaflo system includes techniques for dewatering slurries. Thus, for example, 20-40,000 ton cargoes of slurried iron ore are pumped aboard Marconaflo equipped vessels at dockside in San Nicolas, Peru and in New Zealand, where there are no harbour facilities, similar quantities of slurried iron sand concentrates are pumped about two miles from shore to a monopoint mooring buoy and thence into the ship. In both cases, the cargo is rapidly settled, compacted and dewatered. Upon reaching its destination, the dry cargo is either rapidly reslurried and pumped ashore to holding ponds or discharged dry by means of the more conventional clamshell. Capital, but especially operating costs, are low.

B. Waterborne transportation

In the general case, waterborne mineral transport is more a specialized branch of the shipping industry than a direct adjunct of the mineral industries. That is, the typical mineral producer arranges directly or through a broker to deliver large or small mineral shipments to the port or some other transfer point, whereupon all further arrangements are up to the ship brokers and carriers. Nevertheless, considering the potential savings that can often be realized from more careful attention to transportation as a whole, it is extremely helpful for any mineral producer whose product travels by water to have at least a basic understanding of this form of transport. This is even more true today, since such knowledge may be of great use in planning transportation facilities and systems for which the producer has a more direct responsibility.

Waterborne transport has three components - the seaborne or oceanic trade, the coastal trade or cabotage, and the inland waterways. They are taken in the order listed and their key features briefly described.

Ocean transport

Seaborne transportation services are of two types. There are the liner or conference operators whose vessels travel between specified ports according to published schedules. These carriers are organized into a complex series of regional conferences which establish schedules, types of service, rates and virtually all other details. These carriers lift the majority of the world's "break bulk" (less than shipload) cargoes. The second category of carrier are the tramp (charter) operators. These carriers are, as the name implies, independent operators moving when and where business is to be found. Each voyage or cargo is arranged under an individually negotiated contract or charter.

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There are, of course, many varieties of charter ranging from the single trip between two ports to exclusive use of a vessel for a period of weeks, months or even years. More and more in the bulk mineral trades, specialized vessels such as giant ore/oil carriers are built, operated, and finally scrapped under a long-term charter agreement often arranged to service one particular mining project.

Liner (conference) operators must, by their very nature, be able to lift the varieties and quantities of cargoes available at their scheduled ports. They tend to favour the multi-deck, general purpose, dry cargo vessels epitomized by the now obsolete and disappearing 2,000-ton Liberty class ships. This kind of vessel today is designed and equipped to handle palletized and containerized cargo, loose items, bulk cargo (such as concentrates or grain dumped loose into a hold or carried in drums or bags), refrigerated goods and liquids. It is not surprising, therefore, that the design of these vessels may be very inconvenient in so far as the special requirements of mineral cargoes are concerned. This is particularly true as regards efficient loading and discharge, although on some vessels stowage of bulk mineral cargoes can also be a serious problem.

There is a growing propensity in all aspects of the tramp or charter trades toward the use of more specialized ships designed for maximum efficiency in handling and carrying bulk cargoes. The ultimate example, of course, is the giant oil tanker (up to 480,000 tons capacity). Less well known but no less impressive are the bulk ore carriers and the multipurpose carriers; ore/oil (o/o), ore/bulk/oil (obo), ore/slurry/oil (oso) to name a few. With these vessels, everything is done to maximize efficiency and vessel utilization. Among other factors, they are designed for rapid loading and unloading and handling and rates of 15-20,000 ltpd are commonly specified in the charters. The kind of shoreside bulk storage and material handling equipment required for this type of service needs no elaboration here.

Of potentially great importance to the mineral producer is the fact that these large, 150,000 dwt and up, specialized bulk mineral carriers need a considerable depth of water. Draft requirements of 45 feet and more are common today. It is interesting, therefore, to note that there are only eight European ports capable of receiving vessels of over 200,000 dwt. There is none in the United States. The one other port that can handle vessels of this size apparently is Tokyo. A few new ones are under development or are being planned for the specific purpose of handling minerals.

Another specialized form of ship that is expected to have a noticeable effect upon the mineral industries is the large ice-breaking bulk carrier. The 1969 and 1970 voyages of the converted 118,000 ton Manhattan demonstrated the feasibility of such services.

The subject of maritime shipping is of enormous scope, and there is no time here to do more than merely to introduce the subject. This has been done,

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however, because by far the majority of mineral producers are almost completely unaware of the immense range of possibilities open to them in maritime shipping, and there has been a significant adverse economic effect upon mineral developments throughout the world as a result.

Coastal and inland waterway transportation

These two categories of waterborne transport are taken together for convenience because the line of demarcation between them is growing less. In fact, the focus of this discussion is upon certain new technologic developments of special importance to mineral producers which are helping to further erase the distinction.

Given sufficient room, available only on the large inland lakes or in coastal waters, even the largest oceanic carriers can be used assuming the economic justification exists. Likewise, there has recently been developed a whole class of modern coasting vessels which are nothing more than smaller versions of their larger oceanic cousins, specially designed however for the type of conditions and cargoes encountered in the coastal and some riverine inland trades. These will not be discussed further, not for lack of interest but because of space and time limits.

The most generally employed vessel in both inland and coastal traffic is the barge, usually unpowered, together with independent prime movers, the tugboat or towboat. Barges are basically flat bottomed, more or less blunt ended, usually shallow draft vessels used for carrying goods largely in bulk. They come in all sizes, and today, in a variety of shapes.

Barging developed on inland waterways and spread to sheltered estuaries. Only recently have the marine engineers designed deepwater barges, and the transition from these to oceanic barges is small. Indeed, late stage testing of oceanic towboats and barges is quite well advanced and they should start appearing in commercial service within the next few years.

As more attention has been given to barges, a number of developments have taken place. The technique of pushing rather than towing barges has now been well developed which means much larger tows can be managed in much more difficult waters. One practice, becoming very widespread, is to design a tow using special component barges so that the "hull" configuration of the tow resembles that of an efficient ship. In other words, a tow comprised of, say 24 barges arranged four across and six long will use a lead set of four barges having raked hull shapes so that when they are joined together they give the whole tow a proper bow. Blunt ended barges will be butted together in the centre of the tow and at least one notched stern barge is at the aft end for the pusher tug to engage. In many inland cases, tows and their motive units dedicated to a specific service are specifically designed to break into one to three pieces that exactly fit the locks to be traversed. The list of

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developments is almost endless. Remotely controlled bow rudders are being used to improve tow control and Kort nozzles, propellers in sleeves, enable boats to ply very shallow waters. Towboats with adequate power to handle almost any conditions are now available.

The developments just described have had a significant impact upon inland and coastal traffic, but recently a further development of potentially great importance to the small and middle size mineral producer appeared. This is the LASH (Lighter Aboard Ship) or BOB (Barge on Boat) system.

LASH or BOB is, in retrospect, a simple concept. Specially designed high-speed oceanic carrier vessels and a fleet of barges are employed. The barges, by ones and twos, are scattered along an area of coast and inland waterways where deepwater ports and adequately equipped harbours with heavy lift cranes and similar docking and loading facilities are not available. The barges are loaded and proceed, powered by tugs or other prime movers, to coastal rendezvous points - a small harbour, a sheltered river mouth, or just lying off the coast. The carrier vessel comes by on a predetermined schedule, picking up the loaded barges and dropping empty ones to be loaded. The carrier proceeds to the delivery area and reverses the process. The loaded barges are dropped to be delivered to the consignee who may be well inland and emptied barges are picked up for return to the producer.

There are several different designs of LASH vessels. One can carry 73, 400-ton capacity barges. It uses a 510-ton travelling shipboard gantry to handle the barges and can unload and reload its full barge complement within 24 hours (a 15 minute cycle per barge). A second type, also in service, uses a 2,000-ton capacity hydraulic elevator built into the stern to handle its 38, 850-ton capacity barges. An increasing number of LASH vessels of various designs are coming into use.

C. Air transport

Although not usually included in discussions of mineral transportation, the use of aircraft for carrying mineral products is not that uncommon and there is some evidence to suggest that the frequency of use is growing, albeit slowly.

Obviously, only high unit value commodities can be so carried although the costs of air transport are not as great as they once were thanks to improvements in aircraft design and management. Tin concentrates in Brazil, high grade silver-lead concentrates in Canada, and the three year copper wirebar airlift from Zambia to Dar es Salaam, Tanzania are examples that come readily to mind.

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Aircraft come in all sizes and capacities up to a current maximum for civilian fixed wing planes of about 118 short tons of payload (the Boeing 747 cargo plane). Most commonly used in heavy service such as mining, is the Lockheed Hercules turboprop freighter having a payload capacity of about 24 short tons or an older, better known counterpart, the DC-3, also known as the Dakota or, in its military version, the C-47.

Payload capacity is determined by several variables. Distance is one. The longer the trip, the more fuel is needed, hence the less cargo that can be carried. Length of runway is another and this in turn is further affected by the nature of the runway surface, altitude and ambient temperature. Payload capacity varies inversely with altitude and temperature since air density, hence aircraft lift capability, decreases with increasing altitude and temperature. Conversely, wind speed increases lift, therefore payload.

Most vehicles are characterized by a volumetric or cube limit and a weight limit to the amount of payload that can be carried. In aircraft, a third limitation is frequently encountered, the structural limit. This happened in Zambia, for example. Copper wirebar are heavy and therefore represent a concentrated or point loading on a surface such as the floor of an aircraft. The earlier Hercules planes were not designed for this.

Capital costs of aircraft can be quite high. The Lockheed Hercules cost about $4.0-$4.5 million each. Basic terminal facilities are required including control and maintenance, and usually some kind of storage space, in addition to the runway which can vary from a laterite and gravel surface suitable for a DC-3 or similar aircraft to a high strength, hard surfaced, very sophisticated runway suitable for jet aircraft. Operating costs vary greatly as might be expected, the first determinant being the type of aircraft. The Zambian copper airlift which got poor load factors (planes only operated part of the day) averaged about 11.1¢/ton-mile over a 845-mile run but the Boeing 707 jet is reported to be able to achieve something on the order of 4.0-4.5¢/ton-mile with an 80 per cent or better load factor.

D. Applicability of alternative modes of transport

This section of the paper draws upon the foregoing brief outlines of the more important characteristics of the various modes of transport to discuss the various factors used in evaluating and selecting a mode or combination of modes to be utilized in a specific situation.

There are five major variables, of which three are fundamental, two minor variables, and a number of assorted, largely economic considerations so varied as to defy any sort of aggregative classification. They are listed below. The following discussion is organized around the more significant factors with the minor variables introduced as appropriate.

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Volume

Distance

Time

Continuity of operations and anticipated longevity
Time required to engineer and construct transportation facilities

Commodity characteristics

Physical: particle size and shape, density, dewatering ability
Chemical: soluble, corrosive, reactive

Terrain

Roughness (excessive bank excavation and cut/fill), gradients, spans, curveture, right-of-way width, site engineering and preparation costs, canyon and stream crossings

Climate

Effects on cargo of wetting and/or freezing
Effects on drivers of fog, snow, slippery roads

Existing infrastructure

Characteristics of roadbeds, including strength and overhead clearances of culverts, bridges

Characteristics of harbours and ports: depth, tides, exposure, bottom

Economic factors

Can carry different kinds of cargo?
Backhaul? One way system?
Surplus capacity? Flexibility?
Cube/weight/structural capacity limits?
Route flexibility?
Maintenance, parts availability?
Salvage value of equipment and facilities?
Availability of labour, drivers? maintenance? numbers? training?
Ratio of capital to operating cost and likelihood that the latter will escalate?

Volume and distance

There appears to be no way of separating these two factors from each other so they are considered together. The combination of factors is paramount in the determination of which mode of transport can be used, subject only to the obvious constraint that a steamship cannot operate overland, nor a truck across the ocean.

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Long distance hauls

Railroad v slurry pipeline. Throughout the world, the railroad in one form or another is still probably the most commonly used form of mineral transport over distances in excess of 200-300 miles. Since most minerals travel much further than that before finally being consumed or used to make an intermediate product, it is almost certain that the majority of minerals will be moved by rail during some part of their journey. This statement, however, is predicated upon the availability of a general service railroad, in which case minerals can move in virtually any quantity from the occasional 60-100 ton carload to the 7,500-10,000 ton unit train load. In such cases the producer's problem becomes one of moving the mineral from the mine to the railhead. Again the distance and quantity variables come into play. Small quantities can be carried by truck a couple of hundred miles to the railway, large quantities may justify building several tens of miles of railway spur from the main line to the mine.

If no general service railroad is available and the question is one of whether to build a railroad to service a mining operation, it is quite another matter. The determining factor in this case is the quantity to be moved. The critical point is probably around 1.0-1.5 million tons per year over 70-100 miles. At any smaller quantities and distances, there are several alternatives which will be discussed a little later. For any larger quantities or distances, there are basically only two alternatives - railroad or slurry pipeline.

All other factors being equal, the slurry pipeline is competitive with the railroad under the following conditions: for 1.0 million tpy, over 70 miles; for 2.0 million tpy, over 150 miles; for 5.0 million tpy, over 250 miles; for 10.0 million tpy, about equal at any distance over 300 miles below which the railroad has the edge.

All other factors are never equal. The pipeline requires large quantities of water or other liquid carrier at the upstream end. So far as is known, the cost of having to pump water (a separate pipeline) from the discharge to the loading end of a pipeline has always proved to be uneconomic. Obviously, the fine particle size required for pipelining must be compatible with the further processing and use of the mineral. Clearly, too, the mineral cannot be soluble in the carrier medium. However, if the process flowsheet requires the ore to be reduced to a slurry of fine particles as in flotation, crushing at the mine and a pipeline may be far cheaper than the capital cost of the equipment required to handle and transport coarse material for subsequent crushing.

The pipeline has the advantage over rail in rough terrain. Pipelines can be laid to steep grades (10-19 per cent vs. 1-3 per cent) and much tighter curve radii. Moreover, much cheaper stream and canyon crossings are possible with pipe. When steep grades are present, the potential energy of a slurry at the top is largely converted to pressure energy at the bottom which can then be used to help lift the slurry over the next hill. By rail or road, this same energy is dissipated in braking.

There are always exceptions. The Bu-Craa (Spanish Sahara) 62-mile phosphate conveyor belt is one. Constructed for an eventual capacity of 10.0 million tpy,

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the conveyor was selected in preference to a pipeline because of lack of water and to some degree because pipeline technology was so new in the late 1960s when the choice was made. A wide belt of shifting sand dunes involved foundation soil conditions that could support neither a road nor a railroad able to carry the loads required. Moreover, had truck haulage been used, nearly 300 drivers would have been needed to handle the initial 3.0 million tpy output and almost 700 when full production was attained. No such labour force was or is expected to be available. Incidentally, the capital cost alone of a railroad (disregarding the foundation problem noted above) would have been 50 per cent more than the conveyor system now installed.

A reverse case is the 53-mile long Savage River iron slurry pipeline in Tasmania. The mine is situated only about 50 miles from tidewater and, with an annual production of about 2.25 million tons of pelletizing iron concentrates, it would appear that a railroad was appropriate. In this case, however, the terrain is so rugged that neither truck nor rail nor forms of transport other than a slurry pipeline is economic. That is why the deposit was known for nearly 100 years before it was finally developed and brought into production in 1972.

A final consideration, always present but particularly important in the long-haul situations under discussion, is that of surplus capacity. A slurry pipeline must be designed for a specific capacity encompassing the anticipated production plus some emergency overload allowance. Pipeline operating costs (including the usual heavy capital charges) are extremely sensitive to operation at less than capacity, but there is no surplus capacity in case output were to be expanded at some later date. A railroad, on the other hand, has almost limitless surplus capacity. It is nearly always possible to add a few more cars, run another train or two, or merely speed up trains in order to make more trips. Not so with pipelines. To offset this, however, railroads, as noted earlier, have a very high labour component in operating costs whereas slurry pipelines have practically none. This means that railroad operating costs will continue to escalate in contrast to pipeline operating costs.

The waterborne alternative. The major alternative that is only rarely available to rail or pipeline is waterborne transport. In Guyana and Surinam, for example, alumina is carried by barge a few tens to a hundred miles or so down river and thence in what could be considered a coastal run, to deepwater port facilities at Trinidad. Here the large ocean bulk carriers can load. The draft of these bulkers is too great to pass the sandbars at the river mouths. Smaller oceanic carriers do make the upriver trip, but in many cases their loads are topped off at sea from barges or they too call at Trinidad for this purpose, since they are unable to pass the bars fully laden.

In essence, if a suitable waterway is available, it probably offers the least expensive mode of transport and, unless extensive lock construction or dredging is required, will be a strongly competitive alternative to either a pipeline or a railroad.

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Medium and short hauls

Truck, conveyor, tramway. Unless very large tonnages, on the order of 1.5-2.0 million or more, are involved, the problem of selecting a mode of transport for distances up to 100 miles can be extremely complex, if advantage cannot be taken of an existing system.

Trucks are probably the first mode that comes to mind, particularly if the haul is to a transfer point at a railroad or dock. Trucks are not well suited to large volume transport. A fairly typical truckload is about 40 tons compared to common railcar capacities of 75-80 tons and up. At best, trucks with trailers may manage 60-70 ton payloads per prime mover, but this hardly is impressive when compared to the 7,500-10,000 ton trainload. Assuming no railway or inland waterway is available, the effective distance volume limit for trucks is probably on the order of 250-300 miles and up to 500,000 tons per year. In this type of situation, the truck is virtually impossible to beat assuming road surfaces, gradients, curve radii, strength of culverts and bridges, and drainage are satisfactory.

It was length of haul and roadbed conditions required for trucks, however, that led American Gilsonite Co. in 1957 to install its 72-mile, 6-inch diameter, 400,000 tpy capacity gilsonite pipeline from the mine in Utah to the refinery in Colorado. The pipeline simply passes over the mountain range whereas trucks would have had to go around. The $2 million capital cost of the pipeline was recovered in two years from the differential savings over truck haulage.

Conveyors and aerial trams. At haul distances of roughly 50-60 miles or less, conveyors and aerial tramways start to become serious competitors with road haulage, particularly for quantities of 1 million tons per year and upward. From the earlier discussion, it must also be apparent that under the right circumstances (primarily large volumes of material) a slurry pipeline may also be a strong contender.

The aerial tramway has one great advantage. It can be used in much rougher country than any of the alternatives. On a bicable tramway, for example, an unsupported span of up to a mile can be used to carry material down a steep drop (as at Ertsberg in West Irian, Indonesia) and/or across a wide canyon where even a pipeline crossing would be out of the question. The capacity of a tramway is limited, however, and the ceiling would probably be at around 4 million tons per year. Even then, there would have to be decided advantages arising from bad terrain conditions or some other cause to justify the choice. Moreover, in cases where the tramway and the belt conveyor are more directly competitive, the former has virtually no surplus capacity whereas, like a train, the conveyor does. It can be speeded up, choke fed, etc.

In the case of the expansion of the Reynolds Metals bauxite transport system in Jamaica in the early 1960s, the decision was taken to use a conveyor despite the fact that the company heretofore had depended upon two operating aerial ropeways, a monocable and a bicable. The most important reason for the choice appears to have been the lack of surplus capacity of the ropeway method in view

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of anticipated further expansion. Moreover, tramway operating costs would have been considerably above belt operating costs although capital requirements were generally comparable.

In the Reynolds case, the best possible rail haul would have been 14 miles with bad grades compared to the conveyor's 6.5 mile route. Trucks were also considered, but here again, even accepting 6 miles of 5 per cent downhill loaded grades (rather extreme), the route length was 10.8 miles and climatic conditions, rain and night fogs, would have limited the number of driveable shifts and increased the hazards.

From the few examples introduced into the foregoing discussion of the effects of volume and distance upon mode selection, it is already clear that a number of other variables must be considered in evaluating the applicability of of methods of transport and selecting from among the apparently viable alternatives. The balance of this paper will be given over to reviewing these other factors.

The time factor

Next to quantity and distance, time, both in the sense of continuity as well as longevity, is the major factor in mode application and selection. For maximum economies a slurry pipeline requires a large volume of material continuously passing through it over a long period of time, probably 10-15 years. A special consideration here, although properly a factor in its own right, is the salvage value of a pipeline which, save perhaps for the pumps and a few surface items, is nil. The capital investment, a large one, must be amortized. A railroad, especially if built specifically for the project rather than a general service railroad, is in much the same case except that it has a somewhat higher salvage value.

Both a short project life span and intermittent operation, usually due to climatic conditions, favour truck haulage over most other alternatives. This was the predominant reason for selecting trucks for a 400 km haul from the Kilengwe mine in Zambia to a railroad terminal point from whence the mineral is carried the last 70 km to the smelter. The mine is only worked 10 months of the year, standing idle during the rainy season. It has an anticipated life of 7 years at an annual production rate of about 150,000. Such a deposit, about 1 million metric tons of 11.7 per cent Cu ore, cannot stand the cost of a railway or other major facilities. Incidentally, in this case rail was used for the terminal leg for two reasons. First, the smelter was set up to accept rail, not road, shipments and second, traffic on that last stretch of road was so heavy as to be hazardous.

One other aspect of the time factor that can be quite important is the time required to engineer and construct the transportation facilities. This could readily become a problem in the case, for example, of a concession agreement with a limited initial term and tough termination provisions or where a sales contract has been signed for deliveries to start on a specified date. This was the case

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at a North American coal mine for which a railroad spur would have been the preferred method save for the time required to build it. A conveyor was used instead. The time element for both commencement of sales and under the concession agreement also affected the choice of mode for hauling iron ore at Nimba, Liberia. Terrain factors (grade) were a serious bar to trucks and rail, but other alternatives would have taken too long to construct, hence the conveyor.

Commodity characteristics

The chemical and physical characteristics of the ore or concentrates to be transported also affect the mode to be used. Slurries require fine grain size. Trucks, rail cars, even conveyors can handle quite coarse material and a fairly wide range of particle sizes at one time.

Another physical characteristic that has a bearing, particularly for intermodal transport of slurried material using the Marconaflo system, is the speed and completeness with which the ore or concentrate can be dewatered. This, incidentally, was a telling count against using a slurry pipeline for the Reynolds Metals bauxite project mentioned earlier. Bauxite is very difficult to dewater. The related dry materials case is the propensity, already noted, of some very fine grained, dry material like talc and cement to compact in bins and chutes. The answer here is often a pressure differential system.

Obvious chemical characteristics include solubility, corrosiveness and reactivity. Less well known but a major consideration is the tendency of a number of sulphide concentrates toward autogenous combustion.

Climate

Climatic conditions play an important role in mode selection. The propensity of concentrates to freeze in trucks or railway cars in Canada and other cold climates is an extremely expensive and difficult problem. The effect of fog and rain upon driving conditions in Jamaica has been noted. Cold weather is such a serious problem to one large company trying to develop an arctic iron deposit, that they are seriously investigating an airlift among other remote possibilities.

Terrain

The effect of terrain upon transport, starting with the basic question of whether the commodity moves over land or water, is obvious and has already made frequent appearance in this paper.

The negative effects of rugged country upon railroads and road transport in terms of grades and curvature is clear. Less obvious perhaps are the relative differences in the quantity of cut and fill required for different modes, the costs of cutting and stabilizing high embankments for a road cut as compared with a pipeline trench, or the costs of constructing the number and quality of culverts, bridges, even tunnels for road or rail as compared to the equivalent structures for a conveyor, pipeline or tramway system.

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Another consideration which may be partly physical and partly legal is the available or usable width of right of way. A conveyor, tramway or pipeline needs far less room than a railway or a road.

Existing infrastructure

The obvious effects of an existing infrastructure upon the selection of methods of transportation need not be elaborated upon, especially since it has already been covered in a number of places in the text. Nevertheless, it does deserve specific mention.

Miscellaneous economic factors

As the possibilities begin to narrow down, a host of possible factors and variables must be considered. These, of which only a few examples are very briefly cited here, although important are usually second to the variables already discussed. Every situation is different, however, and under the right circumstances one or more of these could be the deciding factor.

Mixed cargos. Railroad and truck systems can carry ore and concentrate away from the mine and return with personnel, equipment and supplies. Conveyors, pipelines and most (not all) tramways are not suitable for such purposes. Movement of heavy items and materials during the construction phase of a project is not necessarily a reason for selecting rail or road haulage over, say, a pipeline for the movement of ore or concentrate. There is a considerable difference between the road characteristics required to sustain a continued movement of mineral products and the kind of road over which a one-time movement of large machine and other construction items will be made, particularly when heavy earth-moving equipment is available to assist.

Backhauls. In maritime transport and general railroad service, the availability of backhaul cargos can be a very significant economic consideration. The question does not arise (save for comparative system economics) with pipelines, conveyors, or tramways which are all basically one-way carriers.

Surplus capacity and flexibility. The inherent availability of a system to exceed its basic haul capacity or to run economically at less than capacity has already been discussed. This can be a very important factor.

Route flexibility. This is another factor that can be of great importance. Many mining projects actually consist of a number of mines (or deposits) to be worked simultaneously or in sequence with their products passing through one or more central mills, storage or transport terminal points, etc. Trucks have unlimited flexibility, pipelines none, conveyors and railroads some - on occasion.

Labour. The availability of sufficient, skilled labour to operate and maintain a transport system is crucial. This obviously applies to all aspects of a project and will not be discussed further. It is mentioned only because this factor is sometimes overlooked in connexion with transportation.

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Parts and supplies. This too is a familiar subject, no different for transportation than for mine or mill. Diesel engines for instance, are pretty well standard in trucks and railway engines these days, but these engines are no longer anywhere near as tolerant of variations in fuel quality as they used to be.

C. Conclusion

There is no doubt that there has been an amazing change in the technology and techniques of mineral transportation in the last five years. The changes are still taking place. While there have been comparatively few advances that can truly be considered technologic breakthroughs, there have been many, often major improvements on existing technology. Most important of all, however, is the dawning realization of the mineral producers that the entire subject of mineral transport is their concern and the consequent development of management techniques that enable us to approach the subject as an integrated whole.

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X. DEVELOPMENT OF MINING IN BRITISH COLUMBIA

J. B. Evans and J. S. Drake*

A. British Columbia: regional features

British Columbia, the third largest and most westerly Canadian province, with a land area of some 366,000 square miles and a population nearing two and one half million, occupies less than 1 per cent of the earth's land surface and accommodates approximately 0.06 per cent of the world's population, yet the mineral production of this region is significant on a world scale. Currently, mines operating in British Columbia account for the following percentages of total world mineral production: copper at 3 1/2 per cent, lead at 2 1/2 per cent and zinc at more than 2 per cent.

Regionally, more than 90 per cent of British Columbia falls within the Canadian Cordillera and as a result throughout its entire extent, on the North American mainland, on Vancouver Island and in the Queen Charlotte Islands, generally the province exhibits a rough, mountainous terrain. This rugged topography limits the areas available to agriculture, renders land communication difficult and expensive, but provides sites ideally suited to hydroelectric power generation. Approximately one third of the land surface is covered by commercially viable forests and as a consequence the forest industry constitutes the foremost resource base for the region.

Since British Columbia lies some 2,500 miles from the densely populated regions of Eastern Canada, the development of secondary industries has been limited to enterprises related to forest products or to processes involving intensive electric power input. The city of Vancouver, located on Georgia Strait, at the mouth of the Fraser River close to the United States border, serves as the western terminal of virtually all Canadian mainland traffic and is the largest volume export port on the American Pacific seaboard.

Within the Cordilleran region of British Columbia geologic conditions are conducive to the occurrence of mineral and coal deposits, whereas in the Great Plains region, located in the north-eastern corner of the Province, the sedimentary sequence present contains viable petroliferous reservoirs. Recently, Sutherland Brown et al. (1) related the spatial distribution of economic mineral occurrences and coal deposits in British Columbia to five subparallel Cordilleran tectonic belts, the continuous development of which is ascribed to have commenced in the Late Precambrian.

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The introduction of coal mining on Vancouver Island in the mid-nineteenth century coincided with the first permanent European settlement in the Canadian Pacific region. Continuously since that time the exploitation of minerals, fossil fuels and structural materials has played a dominant role in the development of British Columbia. Currently, the quoted wealth generated by the mineral industry, exceeding $300 per capita in 1972, qualifies mining as the industry ranking second in terms of provincial resource development value.

B. The geologic setting

Figure 84, entitled "Generalized Physiographic Regions of British Columbia" indicates the location of the Cordilleran and Great Plains tectonic belts in accord with the Sutherland Brown, et al. (1) interpretation.

Moving easterly from the Pacific Ocean the Cordilleran tectonic sequence is as follows (after Sutherland Brown, et al.):

The Insular Belt encompasses the large, off-shore islands of British Columbia. The southern portion of this belt consists of Triassic submarine basalts overlain by limestones and Jurassic pyroclastic rocks, whereas in the northern portion Paleozoic rocks predominate. Iron and copper skarn, copper-molybdenum porphyry and coal deposits of economic significance occur within this belt.

The Coast Crystalline Belt is essentially a complex of metavolcanic rocks, gneisses and granitoid rocks developed between the Late Jurassic and Early Tertiary periods. Massive polymetallic sulphide deposits are formed within this belt.

The Intermontane Belt is composed principally of Late Paleozoic, Triassic and Jurassic volcanic and clastic rocks. Late Jurassic sedimentation resulted in the formation of appreciable coal reserves within the area of the Bowser Basin, central British Columbia. The presence of extensive copper-molybdenum porphyry and copper-iron skarn deposits results in more than half the presently known copper and molybdenum mineralization of the Canadian Cordillera being located in this belt.

The Omineca Belt is composed of Early Paleozoic and older metasediments and derived gneisses with a relatively small proportion of volcanic rocks present. Tungsten occurs as skarn mineralization. Extensive occurrences of lead, zinc and silver exist as stratiform and vein deposits or, when associated with copper, as massive sulphide deposits.

The Eastern Marginal Belt is made up of Helikian to Paleozoic shelf-type sedimentary sequences containing a high proportion of carbonates. Thrust faults are common throughout this belt. Syngenetic copper and stratiform iron sulphide mineralization occur in this, the least mineralized belt of the Canadian Cordillera. The bulk of the extensive coal reserves of British Columbia and Alberta are located in these sequences.

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GENERALIZED PHYSIOGRAPHIC REGIONS OF B. C.

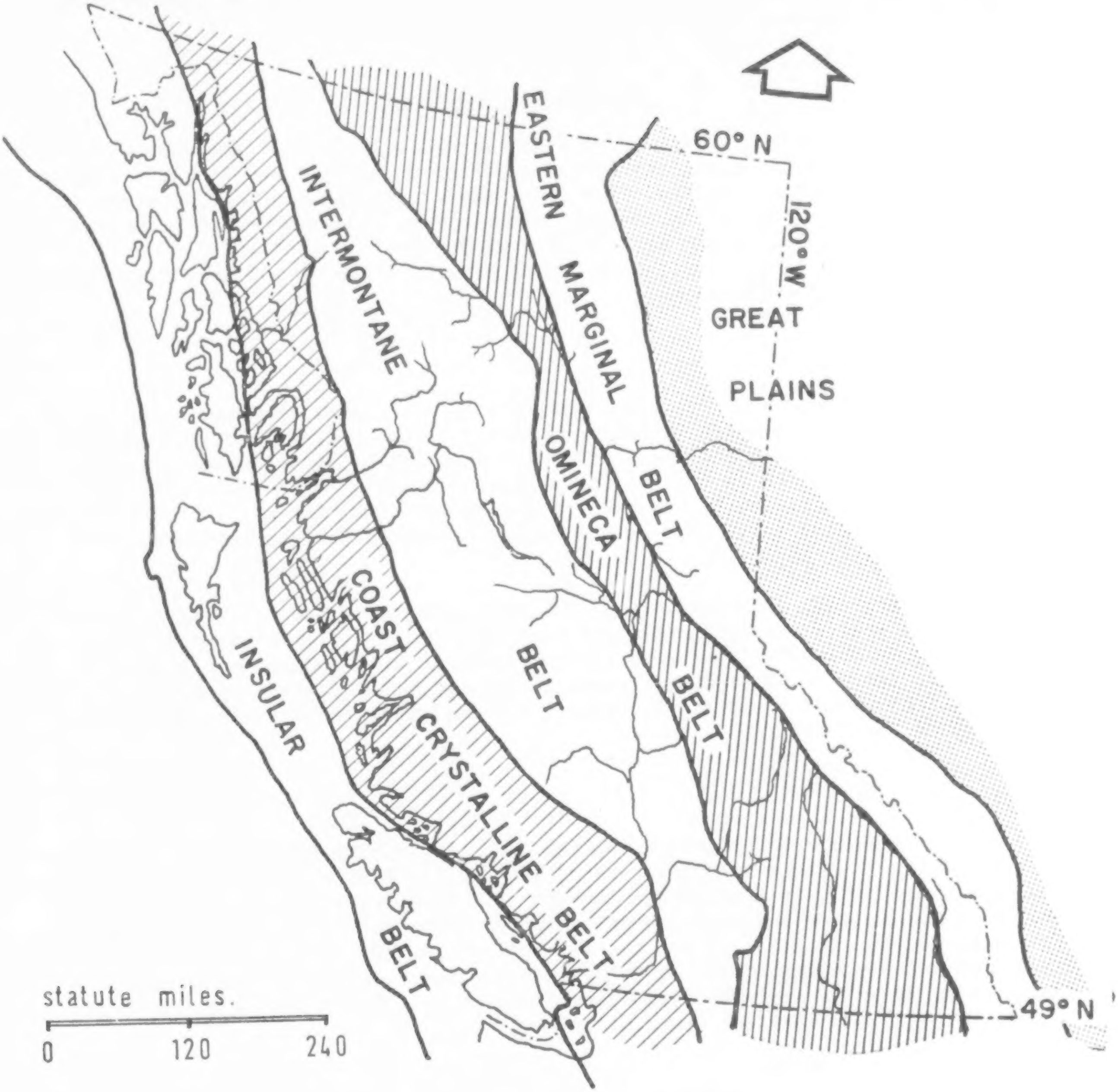
60° N
120° W
49° N
INSULAR BELT
COAST CRYSTALLINE BELT
INTERMONTANE BELT
OMINECA BELT
EASTERN MARGINAL BELT
GREAT PLAINS
statute miles.
0
120
240

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Figure 84

In the north-easterly corner of British Columbia the eastern margin of the Cordillera is flanked by sediments of the Great Plains sequences. In British Columbia, covering a relatively flat land surface some 40,000 square miles in extent, the sequence consists of thick, generally undisturbed sediments ranging in age from Upper Proterozoic through to Upper Cretaceous and contains commercially significant oil and gas reservoirs.

C. Early mining activities

Figure 85, entitled "Mineral Production in B.C., 1900-1972" presents diagrammatically the location, extent and division of mineral wealth generated in British Columbia during specified time intervals. Likewise the volume output of the principal minerals mined in British Columbia over the same period is presented by figure 86, entitled "Volume of B.C. Mineral Production, 1900-1972". Statistical reviews of the British Columbian mineral industry (2) were used in the preparation of these two figures.

The pre-1900 period

Prior to the advent of railroads and related lake ferry services that interconnected inland British Columbia southerly to the United States of America in the 1880s, and through the Rocky Mountains easterly to Eastern Canada in the 1890s, mining activities in British Columbia were restricted to coal mining on Vancouver Island initiated by the fur trading Hudson's Bay Company in the 1830s and to placer gold mining instigated by the Fraser River and Cariboo Gold Rushes of the 1850s and early 1860s.

Immediately rail linkages were established, mining in the Kootenay region of south-eastern British Columbia advanced rapidly: Nelson (silver-copper) 1887; Rossland (gold-copper) 1889; Slocan (silver-lead) 1891; Boundary Camp (copper) 1896; Hedley (gold) 1898; and the Fernie basin coal deposits developed during the late 1890s to provide the fuel requirement for railroads and the many smelters operating in that region.

1900 to 1910

The earlier search for gold along the coast of British Columbia had led to the identification of base metal occurrences, however, in this region the exploration and development of base metal deposits only gained momentum when, in 1906, a custom copper smelter was established on the Pacific Coast at Tacoma, Washington, United States of America. Within the next six years several base metal mines came into production along the British Columbia coastline: Britannia (copper-gold) on Howe Sound; Anyox (copper-gold) on Observatory Inlet; Premier and other gold-silver mines in the Portland Canal district.

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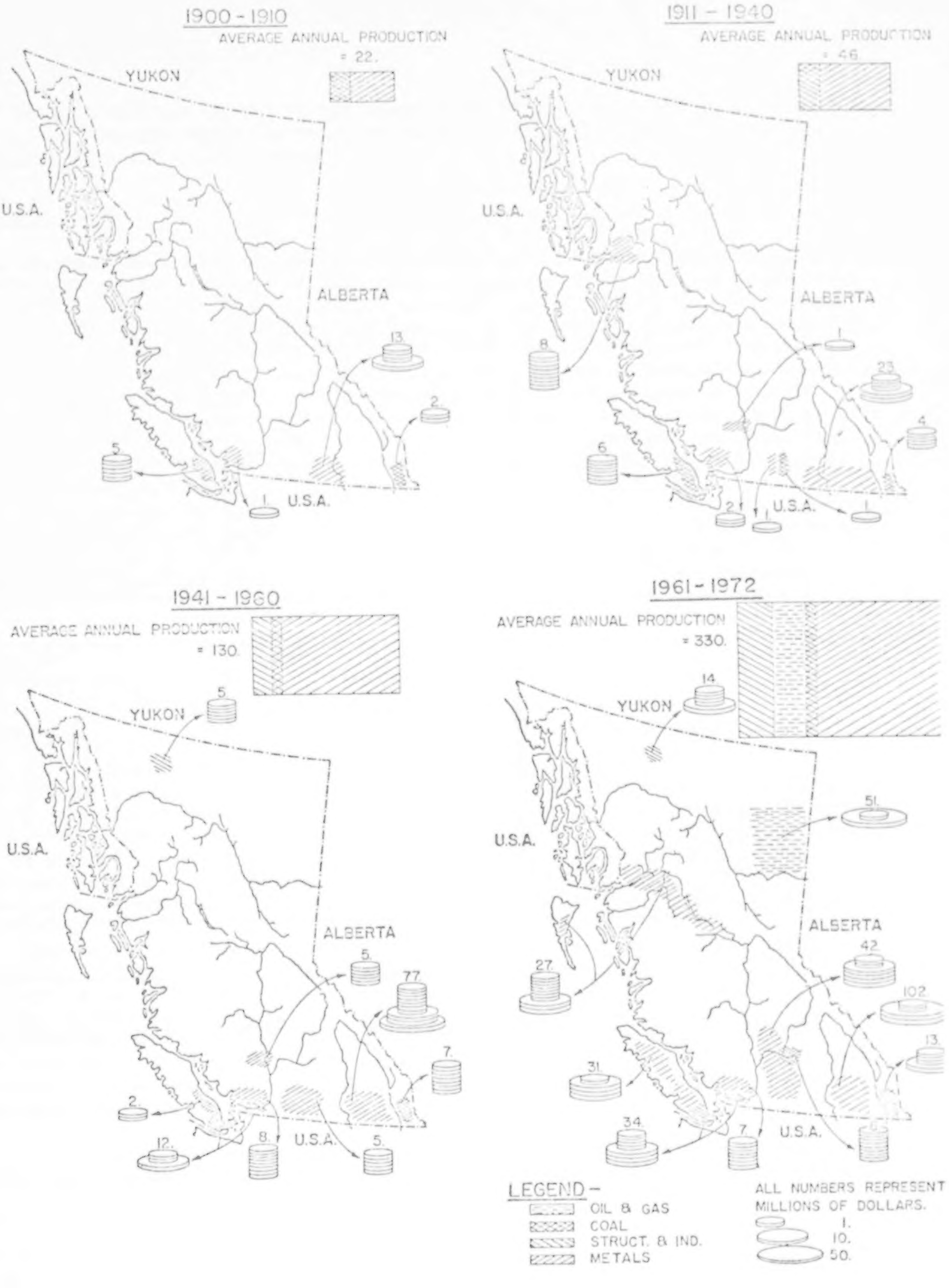
1900 - 1910
AVERAGE ANNUAL PRODUCTION
= 22.
YUKON
U.S.A.
ALBERTA
13
2
5
1
U.S.A.
1911 - 1940
AVERAGE ANNUAL PRODUCTION
= 46.
YUKON
U.S.A.
ALBERTA
1
8
23
4
6
2
1
U.S.A.
1
1941 - 1960
AVERAGE ANNUAL PRODUCTION
= 130.
5
YUKON
U.S.A.
ALBERTA
5.
77
7.
2.
12.
8.
U.S.A.
5.
1961 - 1972
AVERAGE ANNUAL PRODUCTION
= 330.
14
YUKON
51.
U.S.A.
ALBERTA
42
27.
102
13.
31
34.
7.
U.S.A.
6.
LEGEND -
OIL & GAS
COAL
STRUCT. & IND.
METALS
ALL NUMBERS REPRESENT MILLIONS OF DOLLARS.
1.
10.
50.

Figure 85. Mineral production in British Columbia, 1900-1972

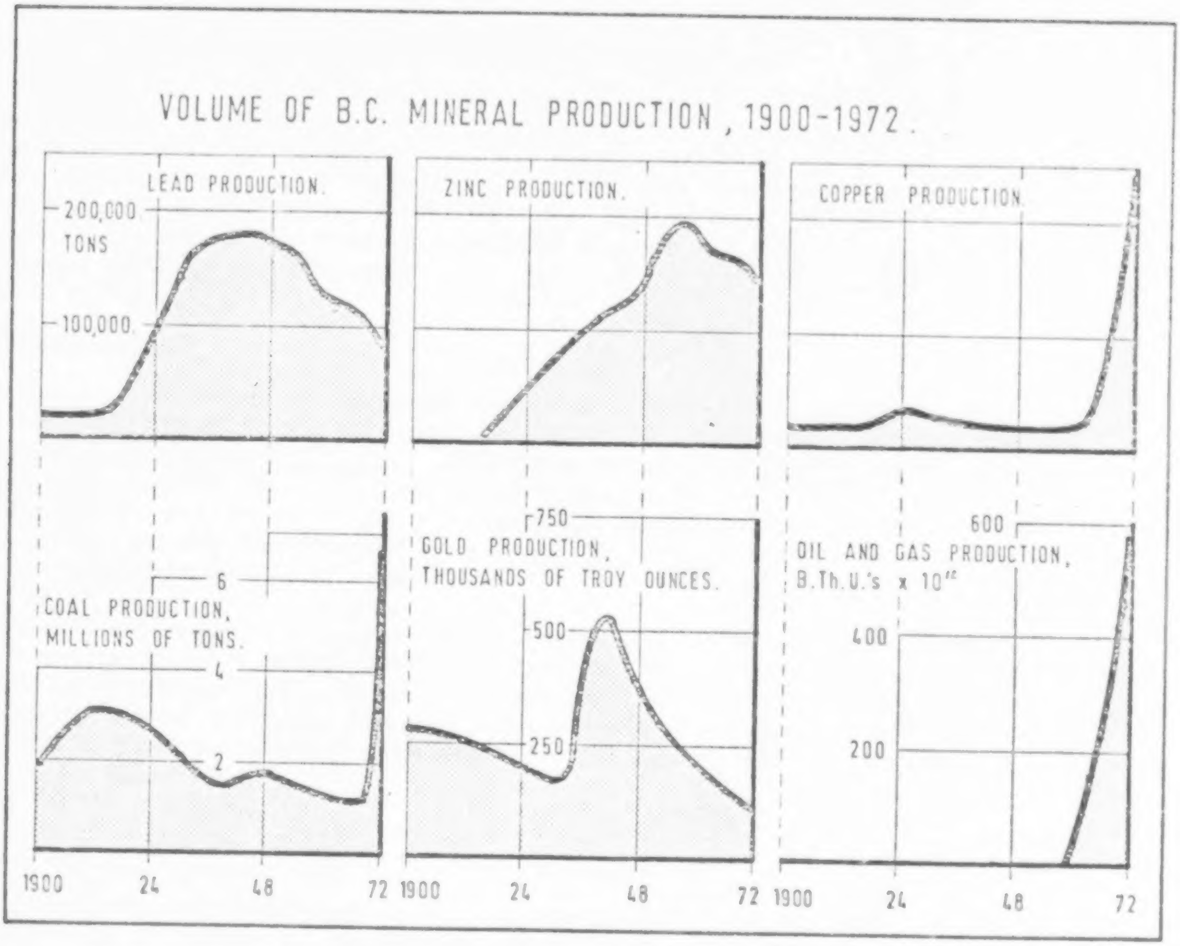

Figure 86. Volume of mineral production in British Columbia, 1900-1972

By 1910 coal mining was firmly established in British Columbia. In that year the output from the mines operating on Vancouver Island and in the East Kootenay district reached 3 million tons.

1911 and 1940

During this 30-year period the introduction of selective froth flotation and the development of electrolytic zinc extraction caused major changes to the total quantities mined in British Columbia: lead and zinc production in the Kootenay district increased tenfold, primarily due to the development of the Sullivan Mine in the 1920s and the subsequent establishment of a fully integrated lead-zinc smelting and chemical fertilizer complex at Trail; copper production increased slightly with Copper Mountain in south-central British Columbia commencing production in the 1920s; whereas over the period, coal production declined to half the rate achieved in 1910. The gold price increase in the 1930s activated many previously known lode gold deposits to yield a peak lode gold output exceeding 500,000 ounces in 1940.

It is interesting to note that despite major technological advances made in the mineral industry and the significant changes in quantities mined in British Columbia in the period 1911 through 1940, no mineral finds of importance were identified in the Province during that 30-year time span. Also, over that interval, mineral production remained relatively static in terms of monetary value per capita and with respect to the proportion of provincial work force engaged in mining.

1941 to 1960

In the 20-year period 1941 to 1960 no new base metal mines of major significance were brought into production, however, pressures induced by World War II accentuated production of particular minerals and resulted in the development of the Pinchi mercury mine in central British Columbia, the Emerald tungsten mine in the Kootenay district and the recovery of tin from the Sullivan mine. With the completion of a mineral resources access road from the coast, in 1952 long fibre asbestos production commenced at Cassiar, north central British Columbia.

Once wartime pressures were removed, production declines resulted: the production volumes of base metals (except zinc) and coal were reduced, the monetary value of minerals produced per capita decreased and the proportion of provincial work force engaged in mining fell to an all-time low figure. When reviewing the development of British Columbia up until the present time (1973), the three-year period 1957 to 1959 stands as the interval within which the mineral industry had its least contribution to provincial prosperity in terms of wealth generated and employment opportunities.

In the late 1950s markets available to British Columbian coal were limited, prevailing base metal prices were low, many base metal deposits were depleted and

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no new base metal occurrence of major significance had been identified within the Province over the last few decades. This period marked the one hundredth anniversary of the Fraser Gold Rush.

Following the initial exploration of the Province by placer gold miners, the passage of a century of mining had produced a record of continuous mineral production achieved essentially from one of two general locations: the relatively narrow southern strip, no more than 100 miles in width adjoining and running parallel to the United States border; and the longitudinal coastal belt, no more than 50 miles in width, including Vancouver Island, the Portland Canal region on the northern mainland and the Queen Charlotte Islands.

D. The modern period

1961 to 1972

The realization that British Columbia displayed the two basic prerequisites essential to the development of a virile mineral industry, namely, a geologic environment conducive to the occurrence of economic mineral deposits and an established, stable legislative system, became apparent to many international mining companies in the late 1950s and early 1960s.

This realization coincided with the identification of the Intermontane Belt of the Cordillera in British Columbia as a probable host to porphyry-type copper deposits and the introduction of Japanese interests as potential buyers of British Columbian mineral concentrates.

The 1961 opening of the Bethlehem Copper open pit mine in the Highland Valley, south central British Columbia, signified the commencement of copper porphyry mining in the Province.

During the last 15 years, 1958 to 1973, concerted exploration programmes involving trained technicians utilizing modern geophysical and geochemical techniques have been undertaken throughout the Province. In addition to examining areas of known mineralization in detail, and attempting to define extensions of such areas by using aircraft, including helicopters, these crews have mapped and correlated large areas previously considered to be inaccessible.

The increase in exploration effort in British Columbia is exemplified by the following tabulation. This tabulation presents a summation of annual exploration expenditures incurred by mining companies operating in British Columbia during the period 1955 to 1973, as reported by the British Columbia and Yukon Chamber of Mines (3):

Year	Amount
1955	$3 million
1960	$4 million
1965	$15 million
1970	$36 million
1972	$38 million

Since 1961 many porphyry-type copper-molybdenum near-surface deposits have been identified, primarily on the mainland in the Intermontane Belt, but also on Vancouver Island in the Insular Belt. Currently, 1973, in British Columbia there are nine open pit operations mining and processing porphyry-type mineralization to yield a total milling capacity exceeding 200,000 tons per day. These operations account for approximately half of the provincial mineral wealth and represent more than 3 per cent of the total quantity of copper mined throughout the world today. Collectively these mines operate at mill head grades equivalent to around 0.5 per cent copper content and report open pit reserves that aggregate in excess of 8 million tons of contained copper content.

In respect to coal mining, immediately following World War II production declined to approximately 1 million tons per annum until 1970, at which time Japanese smelters entered into long-term (14-year) contracts to purchase coking quality coal. The provincial coal production for 1972, derived from one underground and two strip mines located in the Fernie basin, approached 7 million tons.

Following upgrading treatment at the mine site, this coal is moved by unit trains hauling 8,500-ton payloads over distances of some 700 miles to seaboard bulk loading terminals located at Vancouver. Currently two of these terminals each have the capacity to load ships up to 120,000 tons capacity at loading rates greater than 2,000 tons per hour.

A recent survey (4) reports the coal reserves of British Columbia as below:

Measured	7.3 x 10^9 tons
Indicated	11 x 10^9 tons
Inferred	41 x 10^9 tons

With regard to oil and natural gas, during the 1960s the reservoirs in north-eastern British Columbia were brought into production. Currently this production, at an annual valuation exceeding $110 million, represents 15 per cent of the provincial mineral wealth.

E. Monetary and employment patterns, 1900 to 1972

Figure 87, entitled "Reported Monetary Value of B. C. Mineral Production, 1900-1972" and table No. 9, entitled "Relative Importance of Mining to the Province of British Columbia", have been derived from published statistical reviews (2).

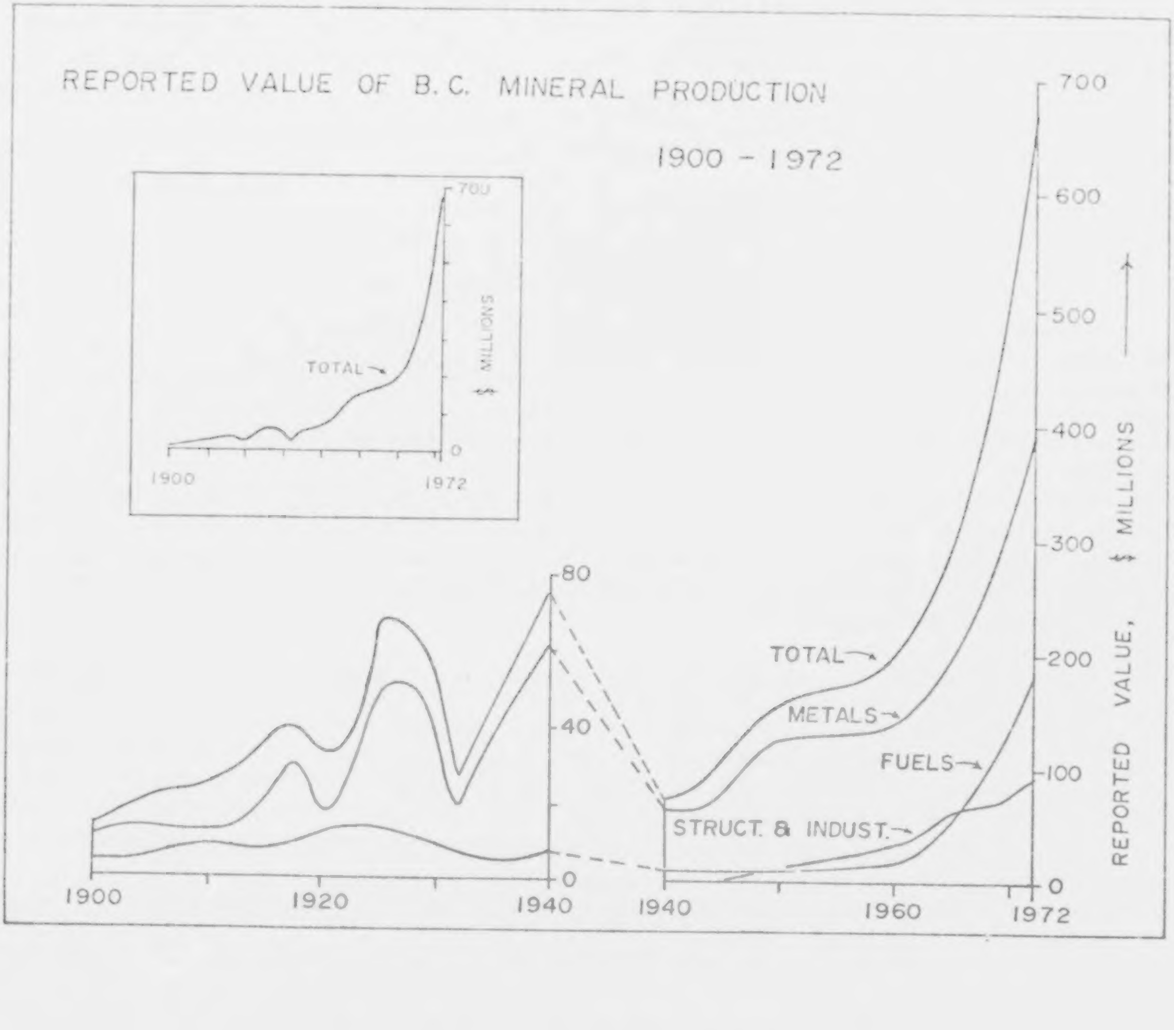
REPORTED VALUE OF B.C. MINERAL PRODUCTION
1900 - 1972
TOTAL
$ MILLIONS
700
0
1900
1972
80
40
0
1900
1920
1940
1940
1960
1972
TOTAL
METALS
FUELS
STRUCT. & INDUST.
700
600
500
400
300
200
100
0
REPORTED VALUE, $ MILLIONS

Figure 87

Table 9. Relative importance of mining to British Columbia

Year	Total population	Value of mineral production per capita a/ (dollars)	Assumed available work force b/	Reported mining company employment c/	Work force employed in mining (percentages)	Mining employees in coal mining (percentages)
1901	150 000	130	50 000	8 000	16	62
1912	400 000	80	110 000	11 000	10	58
1920	500 000	70	125 000	10 000	8	70
1938	800 000	80	200 000	16 000	8	17
1951	1 200 000	150	300 000	18 000	6	13
1961	1 600 000	110	400 000	11 000	3	12
1971	2 200 000	240	550 000	18 500	3 1/2	8

a/ Actual value reported for the particular year.

b/ Provincial male population 18-65 years of age.

c/ Including only persons paid directly by mining companies.

These two exhibits suggest that the following trends have taken place with respect to the mineral industry in British Columbia over the last 70 years.

(1) Subsequent to 1920 metal mining played the dominant role in the mineral industry; however, the exploitation of fossil fuels, the dominant segment of the mineral industry prior to 1920, is becoming increasingly important in the 1970s.

(2) Assuming that each mining company employee causes the employment of three additional persons in service capacities within the Province, then early in the 1900s the prime industry was mining. At that time mining (in particular, coal mining) provided a livelihood for more than half of the provincial population, whereas today, assuming the same employment multiplier factor, the mineral industry provides a livelihood for nearly 15 per cent of the provincial population.

(3) Following World War II when the provincial population reached 1 million the production of structural materials became significant, and today this segment of the industry is represented close to each major town.

(4) In terms of per capita wealth generated and proportion of work force employed, the provincial mineral industry remained relatively static for the period 1912 to 1940.

(5) The last decade, 1962 to 1971, has witnessed a mining industry growth pattern unparalleled in British Columbia during the preceding 60 years. Preliminary statistics available for the year 1972 suggest this trend is continuing with an annual wealth generation reported to exceed $300 per capita.

The preliminary 1972 British Columbia mineral industry output is reported to be:

Total metals	$399.4 million
Non-metallics (mainly asbestos)	29.9 million
Fuels - coal	74.9 million
oil and gas	110.4 million
Structural materials	59.1 million
Total for 1972	$673.7 million

The identification of the Intermontane Belt of the Canadian Cordillera as a favourable host to porphyry-type copper mineralization, together with the introduction of Japanese mineral concentrate buyers in the early 1960s, caused a redistribution of mining activity within the Province. Where previously mines were usually restricted to a relatively narrow strip along the southern and western provincial boundaries, over the last decade a strong trend has developed that locates new mines in a broad, northerly trending central zone, i.e. within the Intermontane Belt.

As a result of this redistribution of mining activity, appreciable quantities of mineral concentrates are now being transported to Prince Rupert on the Pacific Coast via a trans-provincial railroad constructed more than 50 years ago as an access route essential to the development of the northern half of the Province.

In British Columbia, following the inception of porphyry copper mining in 1961, 15 new mines have been brought into production with a combined annual mill throughput capacity of approximately 90 million tons. All but three of these new mines are operated as open pits.

The aggregate capital outlay required to equip these 15 new mines to production is estimated to have been in excess of $850 million, and the total exploration and mineral property acquisition expenditures met by mining companies searching for mineral deposits in British Columbia during the period 1961 to 1973 are judged to have exceeded $200 million.

F. The outlook for the future

Table 10, entitled "Active Mining Operations in British Columbia, 1973", presents the authors' appreciation of the diversity, extent and anticipated life expectancies of mines currently in production, excluding structural material-producing operations.

The tabulation ignores the many mineral and fossil fuel prospects that have been delineated and classified as potential-producing properties by various exploration groups.

It is proposed that the record of past mineral and fossil fuel production, the current production rates and the presence of known but as yet undeveloped mineral occurrences in British Columbia collectively fulfil one of the two basic prerequisites essential to the development of a virile mineral industry; that prerequisite being the existence of a geologic environment conducive to the occurrence of economic mineral deposits.

With respect to the other basic prerequisite, namely, the presence of an established, stable legislative system, it is recognized that mines have operated successfully in British Columbia for many decades in accord with the provisions of various federal and provincial statutes.

Several federal and provincial statutes are undergoing considerable revision at present. The influence of these revisions on the continued virility of the mineral industry of British Columbia cannot be assessed at this time.

Table 10. Active mining operations in British Columbia, 1973

Operating company	Location		Mining method	Geologic setting	Present annual production rate	Current head grade	Anticipated life
Coal							
Fording Coal	Elkford	$50^{\circ}N, 115^{\circ}W$	Strip	Eastern Marginal	1.2×10^6 tons	-	+ 20 years
Kaiser Resources	Sparwood	$50^{\circ}N, 115^{\circ}W$	Strip and U/G	Eastern Marginal	4.5×10^6 tons 1.0×10^6 tons	-	+ 20 years
Oil and gas							
Several	Peace River	$55/60^{\circ}N$; $120/123^{\circ}W$	Wells	Great Plains	Oil 25×10^6 bbl. Gas 425×10^6 mcf.		+ 15 years
Asbestos							
Cassiar Asbestos	Cassiar	$59^{\circ}N, 130^{\circ}W$	Pit	Intermontane	1.2×10^6 tons	8% asbestos	+ 20 years
Lead-zinc							
Cominco	Salmo	$49^{\circ}N, 117^{\circ}W$	U/G	Omineca	0.3×10^6 tons	1% Pb, 4% Zn	+ 5 years
	Kimberley	$50^{\circ}N, 116^{\circ}W$	U/G	Omineca	2×10^6 tons	6% Pb, 5% Zn, 2 ozs. Ag/Ton	+ 20 years
Reeves-MacDonald	Nelway	$49^{\circ}N, 117^{\circ}W$	U/G	Omineca	0.2×10^6 tons	1% Pb, 8% Zn, 2 ozs. Ag/Ton	< 5 years
Teck Corporation	Beaverdell	$50^{\circ}N, 119^{\circ}W$	U/G	Omineca	0.04×10^6 tons	1% Pb, 1% Zn, 18 ozs. Ag/Ton	< 5 years
Western Mines	Buttle Lake	$50^{\circ}N, 126^{\circ}W$	U/G	Insular	0.4×10^6 tons	2% Cu, 1% Pb, 7% Zn, 2 ozs. Ag/Ton	5 years
Nickel							
Giant Mascot	Hope	$50^{\circ}N, 122^{\circ}W$	U/G	Coast	0.4×10^6 tons	0.8% Ni, 0.4% Cu	5 years
Mercury							
Cominco	Pinchi Lake	$54^{\circ}N, 124^{\circ}W$	U/G	Intermontane	0.25×10^6 tons	0.15% Hg (?)	(?) 5 years
Iron ore							
Texada Mines	Texada Is.	$50^{\circ}N, 124^{\circ}W$	U/G	Insular	1.3×10^6 tons	35% Fe, 0.3% Cu	5 years
Wesfrob	Moresby Is.	$53^{\circ}N, 132^{\circ}W$	Pit	Insular	2.5×10^6 tons	40% Fe, 0.6% Cu	5 years
Molybdenum							
Canex-Placer	Fraser Lake	$54^{\circ}N, 125^{\circ}W$	Pit	Intermontane	9×10^6 tons	0.16% MoS_2	+ 20 years
Copper							
Anaconda Canada	Howe Sound	$50^{\circ}N, 123^{\circ}W$	U/G	Coast	0.8×10^6 tons	1.2% Cu	+ 10 years
Bethlehem Copper	Ashcroft	$51^{\circ}N, 121^{\circ}W$	Pit	Intermontane	6×10^6 tons	0.5% Cu, 0.01% MoS_2	+ 20 years
Brenda Mines	Peachland	$50^{\circ}N, 120^{\circ}W$	Pit	Intermontane	8×10^6 tons	0.2% Cu, 0.06% MoS_2	+ 15 years
Craigmont Mines	Merritt	$50^{\circ}N, 121^{\circ}W$	U/G	Intermontane	1.8×10^6 tons	1.1% Cu	+ 5 years
Gibraltar Mines	McLeese Lk.	$52^{\circ}N, 122^{\circ}W$	Pit	Intermontane	13×10^6 tons	0.37% Cu, 0.02% MoS_2	+ 20 years
Granby	Babine Lk.	$55^{\circ}N, 126^{\circ}W$	Pit	Intermontane	5.5×10^6 tons	0.56% Cu	+ 20 years
	Greenwood	$49^{\circ}N, 117^{\circ}W$	Pit	Omineca	0.7×10^6 tons	0.8% Cu	< 5 years
Granduc Operating	Stewart	$57^{\circ}N, 130^{\circ}W$	U/G	Coast	2.5×10^6 tons	1.3% Cu	+ 20 years
Lornex Mining	Logan Lake	$51^{\circ}N, 121^{\circ}W$	Pit	Intermontane	13×10^6 tons	0.44% Cu, 0.01% MoS_2	+ 20 years
Noranda Mines	Babine Lk.	$55^{\circ}N, 126^{\circ}W$	Pit	Intermontane	3.5×10^6 tons	0.5% Cu	+ 20 years
Similkameen Mining	Princeton	$49^{\circ}N, 121^{\circ}W$	Pit	Intermontane	6×10^6 tons	0.5% Cu	+ 15 years
Utah Mines	Pt. Hardy	$51^{\circ}N, 127^{\circ}W$	Pit	Insular	10×10^6 tons	0.52% Cu, 0.03% MoS_2	+ 20 years

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XI. OPEN-PIT EQUIPMENT SELECTION AND MAINTENANCE

Frank H. Buchella, Jr.*

In March 1959, Duval Corporation placed in production its first open pit mine. It has subsequently developed four additional open pit operations in the south-west, including the mammoth Sierrita Property near Tucson, Arizona. The mining rates of these operations range from 15,000 tons 1/ to 225,000 tons of ore and waste per day. The total ore and waste mined daily for all five open pits is approximately 350,000 tons. Through development of these operations, Duval Corporation's engineering staff has gained experience and competence in the selection, operation, and maintenance of equipment for open pit mining.

I. EQUIPMENT: APPLICATIONS AND CAPABILITIES

Equipment for open pit mining can be grouped into four broad categories: site preparation, loading, haulage, and field support and service. A listing of the general range of sizes, production rates, and prices for each major equipment type is presented in tables I through IV. The following compendium covers appropriate field applications as well as limitations inherent in some designs.

A. Site preparation

Clearing, grubbing, and earthwork

Specialized machines. This type of equipment generally operates on the principle of leverage uprooting and mechanical breakage of fallen vegetation. It is used in areas where little or no revenue can be derived from the sale of the vegetation removed. Table 11 summarizes specifications and cost for site preparation equipment.

Bulldozers: crawler, with special attachments. Attachments available for this type of clearing equipment are: brush rakes, stump plows, rippers, demolition and multi-purpose buckets, and grapples. Vegetation is usually uprooted, then dozed into convenient piles for removal or burning. The most economical dozing range for this type cf trash cleanup is a distance of 100 to 150 yards.

Drills: jackhammers, airtracs. These units are principally used in the process of building drill roads or in blasting of surface boulders.

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1/ All tons referred to herein are short tons and dollar values refer to US dollars.

Table 11. Site preparation equipment

Equipment	Size range	Production range	Price range (thousand dollars)
	Clearing, grubbing and earthwork		
Specialized machines	325-1,500 hp 50,000-500,000 #	Brush: 40' trees	Special order 100 +
Bulldozers: crawler with special attachments	125-400 hp 30,000-90,000 #	Shrubs: 20' trees	60-180
Drills: jackhammers, airtracs	125-600 cfm 90-125 psig	1 3/4-4" diameter holes 20-120 ft/hr	12-52
Bulldozers: crawler	125-400 hp 30,000-90,000 #	100-350 cu yd/hr per 100 yd distance	60-180
	Rock breakage		
Drills: augers and boring tools	85-265 hp 10,000-35,000 #	3 1/2-8" diameter holes 30-120 ft/hr	12-96
Drills: wagon and airtrac (percussion) with compressor	365-1,200 cfm 125-250 psig	2 1/4-6" diameter holes 20-80 ft/hr	24-130
Drills: large-diameter (percussion)	900-1,800 cfm 125-250 psig	6-9" diameter holes 30-120 ft/hr	120-260
Drills: rotary	125-500 hp 30,000-200,000 #	4 1/4-15" diameter holes 20-120 ft/hr	40-340
Drills: jet piercing (rotary)	3,000°F @ 2,800 fps to 4,300°F @ 5,300 fps 25,000-100,000 #	5-10" diameter holes 10-40 ft/hr	75-175

Bulldozers: Crawler. These units are confined to preparing a level surface to support large rock drills, rock breaking equipment, or loading equipment. The larger bulldozers are generally equipped with rippers and tilt blade for use in building drill roads in rocky or mountainous terrain.

Wheeled dozers, loaders, tractor scrapers, and motor graders and trucks may be used for the same purpose when terrain is relatively flat or has a great amount of alluvial cover.

Rock breakage

Drills: augers and boring tools. There are many drills of this type manufactured or assembled throughout the world. Most are truck mounted and are built to order. In North America, several large drill equipment manufacturers produce standard models. Most of these drills are gear driven through a "Kelly Bar" arrangement.

Production uses for these drills are confined to operation in alluvium or very soft formations where an explosive charge is required to loosen, not break, the material to be excavated.

Drills: wagon and airtrac (percussion) with compressor. The use of this equipment is limited to modest production requirements, normally in mines with low face (bench) heights requiring small-scale blasts for either environmental or production reasons. These drills are finding expanded use in exploration or development drilling. Special adaptations allow drilling to depths in excess of 600 feet. Another specialized use of mobile, carrier mounted, small diameter percussion drills is in secondary blasting and in drilling "reliever" holes in pit blasting.

Drills: large diameter (percussion). Since these drills have a relatively high unit cost per foot, their use is generally confined to rock formations having compressive strengths in excess of 40,000 pounds per square inch. These rigs are self-propelled, mounted either on tracks or on special wheeled carriers. The wheeled carriers are used where a high degree of mobility is required and over-all penetration rates are not critical.

Drills: rotary. The rotary drill is manufactured in a bewildering variety of sizes and models. For the sake of brevity, let it be said that the rotary drill ranges in size from small, utility vehicle-mounted mechanisms utilizing drag bits to large, crawler-mounted capable of drilling 12 1/4 to 15-inch diameter holes to depths of 60 feet in a single tool run. Numerous models capable of drilling inclined holes are available and so designed as to preclude arduous physical labour in operation.

Nearly all drills in this caregory utilize compressed air to evacuate drill cuttings from the bore hole and to cool bit bearings. By far, most of the drilling done in the open pit mining industry is accomplished with roller bits, principally of "Tri-cone" construction. Even though all drill rigs in this category are characterized as having independently controlled down pressure (tool thrust) and rotary speeds, the larger rigs have independent rotation motors coupled directly to the tool string. Most rigs couple machine weight to the drilling tools to maintain thrust forces between 3,000 to 10,000 pounds per inch of bit diameter. The prime power source may be either electric motor or reciprocating internal combustion engines.

Drills: jet piercing (rotary). Jet piercing drills penetrate rock by subjecting a limited surface area of the rock to intense heat, thus causing rock decrepitation or spalling. The high temperature flame is produced at the burner head by the combustion of fuel oil and oxygen. Rotation of the burner is applied externally by rotating the tool string, similar to standard rotary drilling equipment. One unique feature of these drills is the ability to produce controlled chambers within the drill hole. This type of drill has an economic application only in extremely hard, dense rock. Most of these rigs are operating in the Taconite iron range of the Northern United States and in similar operations in Europe.

B. Loading

Loading equipment of various types and capacities is readily available from a number of manufacturers. This permits an operator to select a design which is tailored to a specific loading application. Specifications and approximate capital cost for conventional loading equipment are presented in table 12.

Rock excavation

Power shovels: engine driven. Engine driven machines are used primarily on jobs of short duration, remote location, or where electrically powered or controlled equipment is impractical. Mobility of these machines is good. However, production rates of these machines are hampered by mechanical control devices and mechanical wear. These machines share the advantages common to all power shovels; i.e., high production rates, capability of handling diverse material, and the ability to operate in a generally adverse digging environment.

Power shovels: electric motor driven. Electric shovels have the advantage of unified control systems, efficient utilization of power, fewer energy conversion devices, and faster cycle times.

Power shovels: scoop. Shovels of this type can handle coarse material, clean up the work area, and produce a high rate of production per applied horse power. Units in the 3 1/2- to 8-cubic yard capacity are in use, principally in quarry operations.

Power shovels: backhoe. The major application of the backhoe as a production unit in open pit mining is in the uranium industry. It is uniquely suited to excavate the sinuous, narrow high grade ore zones typical of uranium deposits.

Backhoes are capable of operating on multiple levels and are ideally suited to excavate material at a lower elevation than its track level.

Units which are wheel-mounted on trucks or crawler-mounted on shovel assemblies are readily available. Hydraulic machines are now replacing the obsolescent cable controlled backhoe

Table 12. Loading equipment

Equipment	Size range	Production range	Price range (thousand dollars)
Rock excavation			
Power shovels, engine driven	125-385 hp 150,000-400,000 #	2 1/2-6 cu yd 300-900 t/hr	175-450
Power shovels, electric motor driven	350-1,500 hp 400,000-2,000,000 #	5-25 cu yd 700-3,200 t/hr	375-1,500
Power shovels, scoop	300-700 hp Undetermined	3 1/2-20 cu yd 650-3,700 t/hr	Undetermined
Power shovels, Backhoe	85-335 hp 15,000-375,000 #	3/4-4 cu yd 60-500 t/hr	20-400
Draglines: general	500-2,500 hp 750,000-3,500,000 #	15-35 cu yd 1,000-3,000 cu yd/hr	Undetermined
Front-end loaders, crawler	65-285 hp 18,000-70,000 #	1-5 cu yd 80-400 t/hr	24-110
Front-end loaders, wheeled	65-700 hp 12,000-200,000 #	1-15 cu yd 80-1,200 t/hr	15-250
Bucket wheel excavators	285-2,750 hp Undetermined	3/4-15 cu yd 400-7,500 cu yd/hr	Undetermined
Earthworking			
Power shovels, engine driven	125-385 hp 150,000-400,000 #	2 1/2-8 cu yd 400-1,200 t/hr	175-450
Power shovels, electric motor driven	400-2,000 hp 650,000-3,200,000 #	9-35 cu yd 1,500-6,000 cu yd/hr	425-2,500
Power shovels, backhoe	85-335 hp 15,000-375,000 #		
Draglines: general	500-2,500 hp 750,000-3,500,000 #	15-35 cu yd 1,500-4,500 cu yd/hr	Undetermined
Front-end loaders, crawler	65-285 hp 18,000-70,000 #		
Front-end loaders, wheeled	65-1,050 hp 12,000-250,000 #	1-24 cu yd 65-1,500 cu yd/hr	15-300
Bucket wheel excavators	285-2,750 hp Undetermined		
Scrapers: tractor	150-1,200 hp 50,000-220,000 #	10-60 cu yd Highly varied	35-310
Scrapers: towed or pulled	Requires prime mover 14,000-40,000 #	6-24 cu yd Varied	15-60
Bulldozers	180-770 hp 36,000-190,000 #	50 cu yd/hr @ 440 yd 350 cu yd/hr @ 440 yd	60-225
Dredges	100-10,000 hp Undetermined	10-200 ft depth 20-10,000 cu yd/hr	Custom fabricated

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Draglines: General. Draglines, clamshells, and other nonrigid digging machines are usually found in large-scale strip casting or underwater excavation where precise control of the excavation and disposal points is not required. Draglines of large capacity have been employed in stripping unconsolidated overburden and in mining blasted, friable type ores such as bauxite. The principal advantage of a dragline is that it may dig either above or below its own working level and can work a larger radius than other excavating devices. Draglines can be obtained in three principal configurations: truck mounted, crawler mounted, or walking frame. Truck mounted units are generally engine driven, and the other, larger types, are electric.

Front-end loaders: crawler. These loaders are characterized as being special adaptations of the crawler tractor-bulldozer. Under-carriage and crawler components have been modified for load-carrying functions. These machines can handle extremely coarse material and are very mobile. They have also found limited use as load-carrying devices in areas where manoeuvrability is limited or working conditions preclude auxiliary haulage equipment.

Front-end loaders: wheeled. The wheeled loader has become one of the most versatile machines used in mining. These machines vary in design from hydraulically-controlled attachments on a standard farm-type tractor to the huge, specially engineered units utilized as a primary production unit. Wheeled loaders are ideally suited to situations requiring great mobility and high selectivity or blending. The larger units are well suited to load-carry operations. The economics of wheeled loaders are very sensitive to working conditions. Cost of operations become prohibitive when wet, rocky work areas and coarse material is encountered.

Bucket wheel excavators. Most continuous excavators now in use in open pits are crawler-mounted and have a full revolving machinery deck. Improvements in metallurgy, electric drive, and control design have made it possible for bucket wheel excavators to be used in consolidated, medium to medium hard formations. These machines generally have a much higher productivity and a lower operating cost than shovels of comparable capacity but face several disadvantages. The machine is relatively immobile and requires strict adherence to mining plans. It necessitates large auxiliary haulage facilities and is of high initial capital cost.

Earthworking

Power shovels. The limiting factor on size and productivity for non-strip-casting uses is the relative size of the haulage system being used for disposal.

The principal difference between rock excavating and earthworking shovels is that earthworkers have a larger bucket size and production rate for the same weight and horse power rating.

Power shovels: backhoe. Because of their relatively low production rates, these machines are used only in stripping or exposing ore zones having an irregular upper surface.

Draglines: general. Draglines find greater application in stripping overburden than any other phase of mining. This is due to the ability to operate on levels other than its own, large working radius, and generally lower ground pressure.

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Machines working in stripping operations generally have a higher productive rate than do those working in semi-consolidated alluvium, blasted material, or dredging operations.

Front-end loaders: crawler. These units find limited application in earthworki and are utilized as a matter of expediency.

Front-end loaders: wheeled. Effective utilization of this type of machine is limited to loosely consolidated material or in a load-carry operation not exceeding distances of 500 yards.

Bucket wheel excavators. These units are ideally suited for handling large volumes of material over extended time periods. They require detailed mining plans and extensive disposal systems.

Scrapers: tractor. Scraper operations are characterized by their high degree of mobility, high production rates, and their limited requirements for auxiliary support equipment. Tractor scrapers are available in two basic designs. The two-wheel tractor is powered with an overhung engine and has a trailing, two-wheel scraper unit. The more powerful machine features the same configuration with a second engine and drive train for the trailing wheels. A specialized adaptation to the basic scraper is a ladder-chain loading device which makes the unit self-loading in many instances.

Scrapers: towed or pulled. Towed scrapers are used only for shallow cuts, short haul distances, and controlled dispersion of material at the disposal site. Their use in open pit mining is generally limited to an adjunct role in site preparation and haul road construction.

Bulldozers. Bulldozers are commonly used in conjunction with a loading chute to strip overburden or load from a stockpile. Push-loading distances in excess of 200 yards are not considered practical.

Dredges. Dredges are manufactured in bucket line and hydraulic designs. The hydraulic unit usually pumps excavated material as a slurry to a disposal area.

C. Haulage

Haulage or transportation costs commonly account for some 50 per cent of total mining costs, thus, the compelling need to carefully plan and engineer the transportation system. The principal systems in use today are railway, truck, conveyor, tramway, and hydraulic. Table 13 summarizes specifications and costs for such equipment.

Railway

Locomotives: diesel-electric. Rail haulage can be economically attractive when mine access gradients of 4 per cent compensated or less can be maintained and when over-all one-way haul distances are three to five miles or more.

Table 13. Haulage equipment

Equipment	Size range	Production range	Price range (thousand dollars)
		Railway	
Locomotives: diesel-electric	150-3,000 hp 50,000-400,000 #	4,500-120,000 # T.E.	75-450
Locomotives: electric	75-4,000 hp 25,000-400,000 #	2,500-160,000 # T.E.	50-400
Rolling stock: gondola cars	2-4 axle 30,000-80,000 #	30-80 tons	20-40
Rolling stock: side dump cars	4 axle 45,000-120,000 #	20-50 tons	60-200
		Truck	
Two-axle: 20 to 55 cu yd Struck, diesel	350-700 hp 60,000-90,000 #	30-60 cu yd, heaped 35-75 tons	95-150
Two-axle: 55 to 70 cu yd Struck, diesel	700-1,200 hp 100,000-170,000 #	65-85 cu yd, heaped 85-120 tons	150-225
Two-axle: 55 to 70 cu yd Struck, diesel-electric	700-1,200 hp 90,000-160,000 #	65-85 cu yd, heaped 85-120 tons	175-250
Two-axle, 70 to 110 cu yd Struck, diesel-electric	1,325-2,000 hp 190,000-245,000 #	80-150 cu yd, heaped 140-200 tons	340-460
Three-axle: 70 to 110 cu yd Struck, diesel-electric	1,800-2,000 hp 320,000 #	140-150 cu yd, heaped 180-200 tons	550
Two-axle: tractor-trailer	300-1,200 hp 40,000-140,000 #	20-80 cu yd, heaped 25-100 tons	50-225
Three-axle: tractor-trailer	325-1,325 hp 40,000-280,000 #	20-140 cu yd, heaped 25-180 tons	50-380
		Miscellaneous	
Bulldozers	125-770 hp 30,000-190,000 #	50-350 cu yd/hr @ 440 yd distance	60-225

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Except for the small locomotives in the 25-60-ton class, power is supplied through traction motors to each axle. Total power output of these machines is limited to the rating of its diesel engine. Common practice is to connect locomotives in tandem when greater tractive efforts are required. All modern locomotives are equipped with dynamic braking systems.

Locomotives: electric. Maintenance costs for electric locomotives are usually one-third to one-half that of diesel-electrics. However, a larger capital outlay is required due to the complex overhead electric transmission system. Because of the semi-permanent nature of this electrification system, all electric railway systems require rigid mining plans.

Rolling stock: gondola cars. This type of rail car is used mainly for ore haulage as it requires a separate rotary dumping mechanism. Bottom dump gondola railway cars are sometimes used for transporting crushed ore to a concentrator and for transporting concentrate to a smelter.

Rolling stock: side dump cars. Dump mechanisms on open pit side dump cars are of the integral air/hydraulic ram type. The shear bulk of the dumping mechanism reduces total car carrying capacity and increases rolling stock maintenance. Howeve this type can be used for both ore and waste haulage.

Truck

Haulage trucks are manufactured in a wide variety of sizes, capacities, and drive systems. They range from the smaller units suitable for general haulage over public roads to the massive, plus 200T capacity units requiring assembly at the mine site.

Two axle, 20-55 cubic yard: struck, diesel. Units of this type generably utilize power shift transmissions and torque converters. Final gear reduction to the drive wheels is accomplished through a "SUN GEARED" planetary axle drive. Torqu retarder braking systems are incorporated in nearly all models produced. Suspension systems are either leaf spring, air cushion, or controlled rubber cushion. These trucks are normally operated over grades up to 15 per cent.

Two axle, 55-70 cubic yard: struck, diesel. Operating characteristics are similar to that of the smaller machines. Units in this size range are universally modular designed wherein propulsion components can be removed as a package. Some models are produced in an eight wheel, two axle drive.

Two axle, 55-70 cubic yard: struck, diesel-electric. Power is supplied to the drive wheels via "SUN GEARED" planetaries driven by independent electric motors usua mounted inside the wheel assembly. Electric current is supplied to the motors by me of an engine-driven generator or alternator. Diesel-electric trucks generally exhib higher availability rates and lower maintenance costs than the all-mechanical units like capacity. Grade capability of diesel-electric trucks is equal to or greater th straight mechanical trucks. Diesel-electrics are equipped with regenerative dynamic braking systems, the capacities of which can be selected to fit anticipated haulage conditions.

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Two axle, 70-110 cubic yard: struck, diesel-electric. Actual hauling capacity is generally limited by tire ratings. Tire costs frequently amount to 50 per cent of total hauling cost in trucks of this size range.

Three axle, 20-70 cubic yard: struck, diesel. Production of three axle haulers has been drastically curtailed since the advent of high capacity two axle haulers. Most three axle units of this size now employed in open pit mines were manufactured prior to 1965. Some manufacturers have produced limited numbers on special order.

Three axle, 70-110 cubic yard: struck, diesel-electric. This vehicle differs from the standard diesel-electric models in that the direct current traction motors are not mounted within the wheel assemblies. A single motor is connected by spur gearing to the transverse drive axle in each of the bogey wheel assemblies. General operating characteristics are similar to other diesel-electric haulage units.

Two axle: tractor-trailer. This class of equipment consists of a two wheel, engine overhung tractor coupled to a trailing haulage unit. The trailing unit can be a loaded "slave" bottom or end dump trailer or a self-loading (scraper) trailer. The scraper-type units may have a second engine and drive train mounted between the two rear wheels. The loading mechanism of the scraper may be of the drag type or a combination of drag and elevating ladder bucket type. These units are noted for their high degree of mobility and production rates. They are generally confined to reasonably level hauls.

Three axle: tractor-trailer. These haulers are also manufactured in a very large variety of sizes and performance capabilities. They are produced in end, side, bottom and ejection dumping configurations. The most widely used type in the mining industry is the end-dump design. Trailer-type units have less grade capability than conventional haulers. Power units are principally diesel engines. Drive trains are straight mechanical with manual shift transmissions or coupled electro-mechanical drives.

Conveyor

The earlier concept of utilizing conveyors for transporting materials short distances in and around plants and as adjuncts to loading devices has led to their occasional use as fully integrated transportation systems. Conveyors now carry materials over distances measured in tens of miles and over or through adverse terrain. It is generally recognized that the direct operating cost of a well-designed conveyor system is substantially lower than those for other haulage systems. However, the application of conveyor haulage is restricted because of lack of mobility and the limitation on the size of material handled.

The two basic types of belt conveyors now being used are internal tension, where the driving force is transmitted within the belt itself, and external tension, where the driving force is carried by externally mounted steel cables. The newer installations use the internal tension principle with the driving force being carried by steel cables within the belt.

In addition to custom designed and fabricated conveyor systems, there are numerous pre-designed and manufactured conveyors available. Most manufacturers supply

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catalogues listing pre-designed conveyors, their size and production capacities, and their suitable applications.

A few specialized conveyors utilizing flexible supports and/or drives have found use in open pit mining. A major type of flexible support conveyor is the unitized flight, movable conveyor which is used extensively in conjunction with bucket wheel excavators. This conveyor can readily be moved in a lateral direction without dismantling. Head and tail pulley drives and assemblies are also mobile. A second type of flexible support conveyor in use is the wire rope system. In this design, the load-carrying idlers are attached to lateral cables supported at intervals by stanchions which also support the return idlers. Cable tension is controlled through turn-buckle arrangements at pre-determined intervals. These conveyors are generally utilized in cross-country systems of moderate capacity, traversing hilly or mountainous terrain.

Another specialized conveying system is the power generating or regenerative conveyor. These conveyors are used in decline installations where the gravitational force produced by the descending load exceeds the frictional forces in the conveyor system. The generated power, normally in the form of electrical energy, is dissipate or used in some appropriate manner.

Miscellaneous

Bulldozers. Bulldozers find use as stripping units by pushing overburden to a disposal area. Production rates are reduced drastically as pushing distances increas The development of the U-shaped blade and the practice of using two or more units in juxtaposition have increased the productivity of this equipment. One manufacturer now produces a double unit coupled in juxtaposition, both units being controlled by a single operator. The coupled units can be separated and used singly.

Hydraulic transportation. Systems utilizing hydraulic principles find limited use in open pit mining operations. Some ores and a variety of overburden are amenabl to hydraulic transport. Engineering and design of slurry hydraulic systems are compl and in many cases, final calculation of capacities and operating costs can only be ma through the use of empirical data or data obtained from pilot operations.

The hydraulic system requires rigid control of slurry density, particle size, size distribution, viscosity, and slurry velocity.

Skip hoisting and ore passes. These methods are sometimes applied where large differences in elevation exist within the haulage system.

Ore passes (shafts, inclines, etc.) are, of necessity, a gravity controlled syst They not only serve as an intermediate transportation device, but also as a surge capacity within the haulage system. Ore passes are normally restricted to a single dump station in open pits and require well-designed mining plans.

Skip hoists, either within a shaft or incline or on a pit slope, are limited in production rates. These hoists normally have high operating and maintenance costs and lack flexibility.

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D. Field support and service

Field support and service equipment is an integral part of any successful open pit operation. Investment in equipment for these purposes usually comprises about 15 to 20 per cent of the cost of equipping the mine. Table 14 summarizes the specifications and costs of such equipment.

General

Cranes: utility, 8-15 ton rating. Cranes in this class are usually mounted on a special four-wheel, self-propelled carrier. All wheeled carriers are equipped with lateral support outriggers, either manually or hydraulically operated, and have a full 360-degree swing. Pipe laying cranes in this capacity range have booms generally side mounted on counterweighted crawler tractors. Crawler units of this size usually have limited boom swing. Specialized cranes with work decks or man cages attached are usually truck-mounted and controlled by hydraulic rams. Some of these high lift platform cranes are equipped with an "elbow-wrist" motion boom rather than the extensible type.

Cranes: 15-45 ton rating. Cranes in this size range are mounted on specifically designed carriers, either crawler, railway, or truck type. These cranes can be obtained with either cable or hydraulic controlled boom angle and boom extension. Crawler-mounted units are generally heavier, have a longer boom, and are cable controlled. Nearly all types have full circle boom swing and are powered by gasoline or diesel engines.

Cranes: 45-110 ton rating. Although these units can be either crawler, railway or truck mounted, all are cable controlled. Cranes having capacities greater than 85 tons have limited use in open pit mining. Due to superior mobility, truck cranes are usually preferred in mines - where they are used primarily for assembly and repair work on major equipment. The diesel engine is the common power plant for these cranes.

Hoist trucks. Commonly known as forklifts, these versatile machines are used in a great variety of ways in materials handling and repair and maintenance. They are basically a self-propelled, wheeled vehicle having an adjustable width set of arms which are attached to a hoisting assembly. Hoist assemblies are either mechanically or hydraulically operated and can be tilted fore and aft. Some models are equipped with a rotating fork mechanism.

There are many attachments for specialized applications now on the market. Standard models produced range from small, three-wheel, externally controlled hand trucks to massive, 15 ton capacity, enclosed cab units. Electric driven machines powered by battery or external source are common in the smaller capacities. Intermediate sizes are gasoline engine driven, and the larger units are equipped with diesel engines.

Field service truck: Fueling. Mobile fuel trucks are indispensable in open pit mining. Storage capacity for each truck is determined by the amount of fuel dispensed per work shift. There is a growing trend towards very large capacity units equipped with high pressure, high volume pumps, and quick-connect nozzle arrangements. Trucks so equipped decrease the idle time of other equipment, particularly haulage trucks. Mobility and manoeuvrability of a fuel truck are important when many items of equipment in outlying areas must be serviced.

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Table 14. Field support and service equipment

Equipment	Size range	Production range	Price range (thousand dollar
		General	
Cranes: all, truck and crawler mounted	85-325 hp 25,000-250,000 #	8-110 ton rating 0-40 mph roadspeed	30-300
Explosive trucks: ANFO mixtures	165-325 hp 4,000-12,000 #	8-21 cu yd 6-14 ton	12-32
Explosive trucks: slurried products	165-325 hp 8,000-16,000 #	5-12 ton rating 10,000-24,000 # Expl.	20-60
Explosive trucks: utility	125-285 hp 3,000-5,500 #	1 1/2-3 ton rating	3.5-6
Motor patrols	85-225 hp 20,000-50,000 #	12-16 ft. blade 15-30 mph, max.	30-85
Utility vehicles: all, including hoist trucks, field service and repair vehicles	85-325 hp 2,500-8,000 #	1/2-5 ton rating 0-70 mph	3-12
Water trucks: road sprinkler	185-500 hp 40,000-90,000 #	3,000-16,000 gal (US) 500-1,500 gpm (US)	30-120

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Field service truck: lubrication. As with fuel trucks, these units are a vital necessity in mining operations. Not only do they serve the basic function of lubrication, but their crews perform some preventive maintenance inspections. Most lubrication vehicles are equipped with a full range of dispensers for operating supplies; i.e., lube oils, hydraulic fluids, gear oils, grease, engine coolants (anti-freeze), solvents, and distilled water. Supply dispensers are generally compressed air stub-pumps. Bulk storage capacity is determined, to some extent, by the total load capacity of the truck chassis selected.

Field service truck: mechanical. Units of this type are usually in-shop built on 1 1/2- to 3-T rated truck chassis. These are further described in the section on utility vehicles.

Field service truck: tire maintenance. These trucks consist of a heavy-duty chassis upon which are mounted a flat bed body, a central jib crane of 2,000- to 10,000-pound capacity, a high pressure air compressor, and an enclosed tool crib. These trucks are normally self-contained in that all tools and repair materials are available for tire repair in the field. However, many mines utilize these vehicles for tire change in the field and for returning the damaged tire to a shop area for demounting from the rim assemblies and subsequent tire repair.

Water trucks: road sprinkler. Units are available from manufacturers, but the majority are fabricated at the mine by converting spare or uneconomic haulage trucks to water tankers. Even the manufactured units are derived from a basic earthworking machine. Tank capacities are as varied as the trucks; however, the most used sizes range from 5,000 to 20,000 US gallons.

Explosive trucks: ANFO mixtures. These units are available with gravity, mechanical screw or chain, and pneumatic discharge systems. The gravity-type trucks, requiring a hydraulic jack assembly, find minimal use. Both the mechanical and pneumatic type trucks normally incorporate a metered fuel oil system and some type of volumetric discharge indicator. Some models are equipped with a grinding mill to increase detonation sensitivities and velocities of the ANFO. Standard mixer trucks, such as used in the preparation and delivery of ready-mix concrete, have been also utilized for explosive loading.

Explosive trucks: slurried products. Explosive dispensers of this type are normally custom built to fit the characteristics of a particular slurry mix. Slurries are delivered under pressure to the bottom of a blast hole by means of a flexible conduit.

Explosive trucks: utility. Vehicles in this class are normally standard or slightly modified models of a flat bed truck in the 1 1/2- to 3-T capacity range. They are used to transport men, miscellaneous equipment, detonating cord, fuse and primers. Detonating caps are always transported to the blasting area in a separate vehicle.

Motor patrols. Motor patrols (graders, blades, etc.) are used in haul road construction and a wide variety of clean-up functions. They are indispensable in haul road maintenance. Another important use of the machine is in building and maintaining mine surface water drainage systems.

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Bulldozers. These machines are used in ground preparation and clean up prior to drilling, clean up of throw material from blast and shovel pits, as a push feeder in front-end loader operations, as a ground ripper and prime mover (pusher) in scraper operations, and in construction of haul or access roads in and around the mine Bulldozers are also used to level and maintain disposal areas, dig pump sumps, and a variety of other ancillary tasks.

Utility vehicles. Utility vehicles are used in a wide variety of transport functions. These units can be equipped as personnel carriers, with work cribs, "A" frame boom and winch assemblies, welding equipment, dump bodies, and hydraulic or cable crane assemblies.

Steam cleaner. The facility normally consists of an accessible, well drained concrete slab supplied with power, water, and fuel. Cleaners can be either stationary or portable.

Some mines have made use of small portable cleaners mounted on heavy-duty trucks; these trucks, equipped with water tanks and pump, are used in the cleaning prior to repair of outlying field equipment.

Other

Ambulances

Ballast spreaders

Compactors, both "sheepsfoot" and "grid"

Compressors

Drills, jackhammer, wagon, airtrac, mobile travel

Electric cable reels and spools

Electric cable towers

Fire trucks

Light plants, portable

Pumps

Radio communication equipment

Roadsweepers, self-propelled and towed

Track layers, crawler, rail, and wheeled

Track movers, crawler tractor

Track tampers

Traffic control devices

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In addition to the above items of equipment, most mines maintain a well-equipped repair shop containing many of the stationary bench tools found in machine shops. The extent to which a shop is equipped depends upon the size and scope of mining operations as well as the availability of custom repair facilities in the surrounding area.

II. FACTORS DETERMINING EQUIPMENT SELECTION

The principal factors influencing equipment selection can be grouped into five basic categories.

Tonnage requirements, ore and waste

Loading, transporting, and ground preparation equipment are largely determined by tonnage requirements. Generally, the greater the tonnage requirements, the larger the equipment size selected. Large size equipment is appropriate for a production of about 6 million tons (total) per month, medium size for 3 million tons, and small size for 1 million tons or less.

Topography and property line considerations

Topographical features and property limits influence the location of processing plants and waste disposal areas, which in turn influence the selection of equipment for transporting functions. Length of haulages are considered to be short, if distance is around 1,000 feet; medium, if some 10,000 feet; and long, when 20,000 feet or more.

In the consideration of haulage equipment, anticipated grades are usually categorized as 1-3 per cent, 4-7 per cent, 8-10 per cent, 10-20 per cent, and 20 per cent plus.

The flexibility required of the transportation system is characterized as requiring frequent relocation, occasional relocation, or minor relocation.

Ore control requirements

Working conditions, space limitations, and ore tenor generally govern the selection of loading and ground preparation equipment. Factors to be considered include bench heights where 25 feet is classified as a low height, 35 feet as an average height, and 50 feet and over as a high bench height. Another factor would include the average thickness of the ore zone which is defined as narrow, when the zone is approximately 25 feet; medium, when about 50 feet; and wide, with a zone of 100 feet and over. A third factor includes the mobility required for blending ore and for development work. The range of equipment movements may be classified as frequent, occasional, and seldom.

Physical characteristics of ore and waste

The selection of equipment is also influenced by physical properties of the ore and waste, necessity for drilling and blasting, fragmentation, loading characteristics, and height of blasted material to be loaded.

Climatic conditions

Climatic conditions are important because extremes in temperature, rainfall, and snowfall may dictate the need of special features or types of equipment.

A. Equipment selection

The process by which equipment is selected is basically one of reviewing equipment characteristics and capabilities along with the mining and the anticipated operating requirements and environmental conditions. The single objective of such review is to assure a selection consistent with optimum economics.

A convenient means of matching equipment characterisitcs and capabilities to the mining requirements is illustrated in table 15. In the example presented, the loading and transporting equipment is selected for a typical open pit copper mine located in the south-west United States. Mine planning and engineering studies established the following conditions and mining requirements which are used in the illustration:

1. Tonnage of 2,350,000 tons per month.
2. Average distance to waste dumps of 10,000 feet.
3. Seven per cent maximum adverse grade.
4. Limited dump space due to property boundaries, therefore, frequent vertical lifts required.
5. Quartz monzonite of medium hardness.
6. Drilling and blasting necessary.
7. Average fragmentation expected.
8. Bench height of 50 feet.
9. Ore zones of 100 feet and over.
10. Occasional moving required for blending and development.
11. Average annual rainfall of 15 inches.
12. Temperature extremes of 15° to 105°F.

When equipment characteristics are compared to the mining factors, it is determined that the medium tonnage requirement of 2,350,000 tons per month can be fulfilled by the following types and capacities of loading equipment:

Excavators	1,000-3,000 T per hour
Draglines	12-25 cubic yards

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Table 15 **EQUIPMENT SELECTION CHART**

EXAMPLE STATED	MINING FACTORS	LOADING FUNCTION												TRANSPORT FUNCTION								
		EXCAVATOR			DRAGLINE			FRONT END LDR			SHOVEL			CONVEYOR			TRAIN			TRUCK		
		TONS PER HR			CUBIC YARDS			CUBIC YARDS			CUBIC YARDS			TONS PER HR			LOCOM WT (TON)			TONS PER LOAD		
		250 1000	1000 3000	PLUS 3000	2.5 12	12 25	PLUS 25	3 5	5 15	PLUS 15	2.5 8	8 15	PLUS 15	250 1000	1000 3500	PLUS 3500	25 50	50 100	PLUS 100	25 65	65 100	PLUS 100
x	PRODUCTION REQUIREMENTS LARGE _ 8,000,000 TPY MEDIUM _ 3,000,000 TPY SMALL _ 1,000,000 TPY		x			x				x		x			x			x	x		x	x
x	QUALITY CONTROL REQUIR'MT 1 BENCH HEIGHT LOW _ 25' MEDIUM _ 35' HIGH _ 50' & OVER			x		x	x	ELIM				x	x	N/A								
x	2 WIDTH OF AVERAGE ORE ZONE NARROW, 25' MEDIUM _ 50' WIDE _ 100'	x	x	x	x	x	x				x	x	x	N/A								
x	3 MOBILITY REQUIRED FREQUENT MOVEMENT OCCASIONAL " MINOR "	x	x		x	x	x				x	x		x	x			x		x	x	x
x	ROCK CHARACTERISTICS 1 DRILLING & BLASTING REQ YES NO		x	x		x	x				x	x	x	N/A								
x	2 ROCK HARDNESS VERY HARD HARD MEDIUM SOFT	x	x	x	x	x	x				x	x	x	N/A								
x	3 FRAGMENTATION POOR MANY BOULDERS 5-8' AVERAGE SOME " 3-5' GOOD _ FEW " 1-3'	ELIM			ELIM							x	x	ELIM				x	x		x	x
x	4 HT OF MUCK PILES BLASTED 10-20' 20-35' 40' & OVER											x	x				N/A					
x	AREA TOPO & BOUNDARIES 1 LENGTH OF HAUL SHORT _ 1000' MEDIUM _ 10,000' LONG _ OVER 20,000'										N/A										x	x
x	2 MAXIMUM HAULAGE GRADE 1-3% 4-7% 8-10% 10-20% 20% & OVER										N/A						ELIM			x	x	x
x	3 FLEXIBILITY REQUIRED FREQUENT RELOCATION OCCASIONAL " MINOR "										N/A									x	x	x
x	CLIMATIC CONDITIONS 1 TEMPERATURE HIGH _ OVER 80° (F) LOW _ BELOW 0° (F)										x	x	x							x	x	x
x	2 PRECIPITATION LOW _ 0-15"/YR MODERATE 15-50"/YR HEAVY _ OVER 50"/YR										x	x	x							x	x	x
	SUMMATION										6	⑩	8							5	⑧	⑧

Loaders	10-15 cubic yards
Shovels	8-15 cubic yards

However, it is also determined that shovels in the 8- to 15-cubic yard capacities meet the requirements of rock characteristics while excavators, front-end loaders, and draglines do not.

Similarily, it is determined that due to the length of haul of 10,000 feet, haulage trucks and conveyors can be considered but trains are unsuitable. Rock characteristics dictate truck haulage over a conveyor system, unless primary crushing is considered prior to conveying.

Thus, by the process of comparison and elimination, the loading and transporting equipment selected for this particular mining operation would be electric shovels in the 8- to 15-cubic yard range and haulage trucks in the 65 to 100-T and over 100-T capacity range. Table 16 summarizes equipment selection and related capital costs for five examples of different tonnage requirements.

B. Maintenance

The maintenance programme is vital to the success of any open pit mining operation. Repair and maintenance can be expected to account for 40 to 50 per cent of direct mining costs.

The need for careful initial planning often goes unrecognized, and this leads to underestimation of personnel and facility requirements. In a well-balanced mine organization, maintenance personnel will comprise approximately 40 per cent of the total.

Organizational structure, duties and responsibilities as well as lines of authority, must be clearly defined and understood. The organization should include a maintenance staff group with responsibility for control of procedures, conducting inspections, and analysis of cost data. With these responsibilities resting with a control staff, line supervisors are left free to direct shop and field maintenance work and handle personnel matters.

Effective control of work quality, time expended, and materials consumed is basic to an efficient maintenance programme. These goals are believed best achieved by separating the planning, inspecting, and analysis functions from the work implementation function.

Written work orders which initiate work assignments and also furnish feedback data are essential for a well-controlled maintenance programme. A well-organized and executed programme of preventive maintenance is vital and is an important factor in scheduling the department workload.

Proper warehouse practices are also essential to an efficient programme. The required materials and supplies must be available in the proper quantities and at the proper times. Moreover, good warehouse practice assures accurate cost allocation of materials used.

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Table 16: OPEN PIT MINING EQUIPMENT
TYPICAL OF TRUCK HAULAGE OPERATIONS
IN THE SOUTHWEST UNITED STATES

TOTAL MINE CAPACITY & REMARKS	DRILLING & BLASTING NO & TYPE	COST	EXCAVATION-LOADING NO & TYPE	COST	HAULAGE-DISPOSAL NO & TYPE	COST	FIELD SUPPORT NO & TYPE	COST	MISC SUPPORT NO & TYPE	COST	POWER & UTILITIES NO & TYPE	COST	SHOPS & ANCILLARY		TOTAL CAPITAL COST
10,000 TPD ROCK	1. 6" ROTARY RT	140,000	1. 6 CU YD ELEC SHOVEL	410,000	8. 50T END DUMP TRK	800,000	1. D8 w/ RIPPER	95,000	3. 3/4 T PICKUP	12,000	1. 2.5 KV GEN	40,000	48 x 60 SHOP	[illegible]	
NOM S R 1:1	1. 3.5" HAMMER DRILL	24,000	1. 5 " " LOADER	95,000			1. 824 DOZER	90,000	1. 3T FUEL/LUBE	14,000	2000' POWER CABLE	16,000	20 x 40 OFFICE &		
" HAUL 1 MILE	1. 600/900 CFM COMP	26,000					1. 14E PATROL	50,000	1. 3T REPAIR/WELD	8,000	2 MI WATER/GAS	27,000	CHANGE RM	20,000	
									1. 3T W/CRANE	8,000	MISC	20,000	TOOLS ETC	15,000	
SMALL															
300,000 TPM															1,960,000
30,000 TPD. ROCK	2 _ 9" ROTARY RT	300,000	2 _ +8 YD ELEC SHOVEL	1,080,000	15 _ 85T DUMP TRK	2,625,000	1 _ D9 w/RIPPER	135,000	4 _ 3/4 T PICKUP	16,000	3000' CABLE	24,000	60 x 72 SHOP	60,000	
NOM S R 1:1			1 _ +10 YD LOADER	180,000			1 _ D8 w/RIPPER	95,000	1 _ 3T FUEL/LUBE	14,000	1 _ 2.5 KV TRANS	25,000	20 x 40 OFF/CHANGE	20,000	
" HAUL 1 MI							1 _ 824 DOZER	90,000	2 _ 3T REPAIR/WELD	16,000	2 _ 1320/440 "	40,000	TANKS, FENCE, ETC	16,000	
							1 _ 14E PATROL	50,000	1 _ 12T FORKLIFT	30,000	2 MI 2.5 KV CABLE	80,000	TOOLS, ETC	20,000	
					1 _ 7000 G WATER TRK	35,000			1 _ 15T CRANE	45,000	2 MI WATER, GAS, MISC	60,000			
SMALL															
875,000 TPM															5,056,000
80,000 TPD. ROCK	2 _ 9 7/8" ROTARY C	475,000	3 _ 12 YD ELEC SHOVEL	1,950,000	17 _ 120T DUMP TRK	3,825,000	1 _ D9 w/RIPPER	135,000	8 _ 3/4T PICKUP	32,000	2 MI 5 KV LINE	140,000	60 x 120 MAINT SHOP	150,000	
NOM S R 1:1	1 _ 9" ROTARY RT	150,000	1 _ +10 YD LOADER	180,000			2 _ D8 w/RIPPER	190,000	1 _ 3T FUEL/LUBE	14,000	7,000' CABLE	56,000	48 x 60 TIRE/LUBE SH	75,000	
" HAUL 1 MI	1 _ AIRTRAC	65,000					2 _ 824 DOZER	180,000	1 _ 2T COMP TRK	12,000	TRANSF, ETC	115,000	TOOLS	40,000	
							2 _ 14E PATROL	100,000	2 _ 3T REPAIR/WELD	16,000	GAS/WATER	80,000	TANKS, ETC	30,000	
MEDIUM							2 _ EXPLO TRK	50,000	2 _ 15T CRANE/LIFT	80,000			40 x 80 OFF/CHANGE	40,000	
2,350,000 TPM					2 _ WATER TRK	70,000			1 _ 30 MAN BUS	8,000					8,258,000
120,000 TPD ROCK	3 _ 12 1/4" ROTARY C	975,000	4 _ +15 YD ELEC SHOV	3,200,000	20 _ 150T DUMP TRK	5,700,000	2 _ D9 w/RIPPER	270,000	12 _ 3/4T PICKUP	48,000	2 MI 5KV LINE	140,000	60 x 168 MAINT SHOP	350,000	
NOM S R 1:1	1 _ 9" ROTARY RT	150,000	1 _ +15YD LOADER	240,000			3 _ D8 w/RIPPER	285,000	4 _ 3T FUEL/REPAIR	48,000	10,000' CABLE	80,000	48 x 60 TIRE/LUBE	75,000	
" HAUL 1 MI	1 _ AIRTRAC	65,000					2 _ 824 DOZER	180,000	2 _ 2T COMP/UTILITY	25,000	TRANSF, ETC	125,000	TOOLS, COMP	120,000	
							2 _ 14E PATROL	100,000	2 _ 15T CRANE/LIFT	80,000	GAS/WATER	90,000	60 x 80 OFF/CHANGE	110,000	
MEDIUM							2 _ EXPLO TRK	50,000	1 _ 35T CRANE	100,000	PORTABLE POWER &		TANKS, ETC	40,000	
3,500,000 TPM					2 _ WATER TRK	70,000			2 _ 30 MAN BUS	16,000	LIGHT	25,000			12,757,000
200,000 TPD ROCK	4 _ 12 1/4" ROTARY C	1,300,000	6 _ +15YD ELEC SHOVELS		27 _ 150T DUMP TRK	9,100,000	2 _ D9 w/RIPPER	270,000	16 _ 3/4 T PICKUP	64,000	4 MI 5KV LINE	280,000	90 x 240 MAINT SHOP	700,000	
NOM S R 1:1	1 _ 9 7/8" ROTARY RT	200,000		4,800,000			4 _ D8 w/RIPPER	380,000	5 _ 3T FUEL/REPAIR	58,000	21,000' CABLE	168,000	80 x 120 HD WELD/TIRE/		
" HAUL 1 MI	1 _ AIRTRAC	65,000	1 _ +15 YD LOADER	240,000			3 _ 824 DOZER	270,000	2 _ 2T COMP/UTIL	25,000	TRANSF, ETC	200,000	LUBE SHOP	160,000	
							2 _ 14E PATROL	100,000	4 _ 15T CRANE/LIFT	160,000	GAS/WATER	120,000	TOOLS, COMP	175,000	
LARGE					2 _ 7000 G WATER TRK	70,000	1 _ 16G "	85,000	1 _ 85T CRANE	240,000	MISC EQUIP'T	75,000	TANKS, ETC	75,000	
5,835,000 TPM					1 _ 12,000 " "	60,000	3 _ EXPLO TRK	90,000	3 _ 30 MAN BUS	24,000			120 x 180 OFF/CHG	225,000	19,779,000

EXPECTED MECHANICAL AVAILABILITY UNDER NORMAL CONDITIONS

		8 _ 10 YD EXPLO TRK	75 %	12 1/4" ROTARY C	85 %	15 YD ELEC SHOV	85 %	WATER TRK	90 %
		AIRTRAC	70	5 CU YD LOADER	75	50T MECH TRK	85	D9 DOZER	80
		6" ROTARY RT	75	10 " "	75	75-85 " "	80	D8 "	85
		9 " "	75	15 " "	75	85-100 ELECT "	85	824-834	90
10 - 15T CRANE	85 %	9 7/8 " "	75	6 " ELEC SHOV	85	120 " "	80	14E PATROL	90
85 T "	95	9 7/8 " C	80	+8 " " "	85	150 " "	85	16G "	85

FROM L G DYKERS, "MINING EQUIPMENT REQUIREMENT" UNPUBLISHED MONOGRAPH OF DECEMBER 1970, REVISED NOVEMBER 1972

The systematic accumulating and recording of maintenance data are essential. Standardization of maintenance reports is important for the assembly of useful data.

The record system should include all equipment from service vehicles to haulage trucks. Data should be compiled daily from repair reports and entered on individual equipment records. In addition, records should be kept on equipment components. For example, the record on an electric wheel haulage truck would contain data on such components as the engine, generator, wheel motors, wheel motor armatures, etc. Records on such items as drill bits, shovel teeth, wire rope, and other warehouse items are useful in determining preventive maintenance and overhaul schedules as well as equipment availability factors.

Careful planning and forecasting along with appropriate budgetary controls are integral parts of the maintenance programme.

Other important factors for a successful programme are planned equipment shutdowns, standard overhaul procedures, and engineered work measurements. Engineered work measurements should continually be compared with actual labour performance and if necessary, modified to realistic standards for planning shop loading, scheduling, and control.

Finally, functionally designed and well-equipped shop facilities are fundamental to a successful programme.

Separation of service facilities such as lubrication, tires, etc. from major maintenance and facilities is generally desirable. The following features are recommended for the principal shop facility:

1. Generously sized repair bays.

2. Adequate overhead crane capacity.

3. Drive-in bays for haulage trucks, water trucks, front-end loaders, motor patrols, rubber-tired dozers, and crawler dozers.

4. Floor surfaces with proper wear characteristics for high impact and abrasion service. (Metalized concrete is preferable to embedded rails which present safety hazards.)

5. Acetylene, oxygen, compressed air, water and electrical power outlets at centrally located points for flexibility and accessibility.

6. A centrally located warehouse with accessibility to all working areas.

7. A check out arrangement for control of specialized tools.

8. Separate repair areas or bays for welding, machining, automotive, electrical and engine work.

9. An auxiliary steam cleaning and washing area (clean equipment facilitates repairs and reduces housekeeping in the shop).

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A good maintenance programme supported by adequate facilities will maximize equipment availability and productivity, which in turn will result in safe, efficient and low-cost mining.

C. Operating costs

Operating expense is commonly expressed in terms of cost per unit of material moved. This cost generally includes supervision, operating labour, explosives, fuel, lubricants, bits, tires, repair and maintenance labour, repair and maintenance materials, and miscellaneous mine overhead costs. Depreciation, interest expense, taxes and corporate overhead are usually reported separately.

Cost per unit of material moved is the yardstick in evaluating performance. Most mines consider operating costs to be confidential.

Unit mining costs are usually segregated into centres reflecting a specific operational function. Commonly, such principal functions are engineering, development, drainage, drilling, blasting, loading, hauling, roads and dumps and general. Into each of these centres are charged the applicable supervision, operating labour, supplies, repairs, including labour and materials and miscellaneous items of expense.

Cost information gathered in 1971 from four open pit properties in western and south-western United States ranged from 22 cents to 30 cents per ton of material moved. These costs did not include depreciation, interest, taxes or corporate overhead.

A distribution by functions of these costs is detailed in table 17.

Table 17. Mining costs by function expressed as a percentage of the total direct cost

Function	Approximate monthly production	
	6 million tons	1 million tons
	one property	range of three properties
	percentages	
Engineering	1.7	3.0-3.4
Development	1.8	2.3-3.8
Drainage	0.8	0.1-0.7
Drilling	8.3	6.9-10.4
Blasting	15.5	7.0-12.6
Loading	16.4	16.6-22.5
Hauling	41.2	36.0-42.1
Roads and dumps	9.6	10.0-12.9
General	4.7	4.8-7.2

/...

D. Conclusion

In the past 60 years, a dramatic change is noted in material handling equipment. The coal burning steam shovel of the Chino, New Mexico pit, which required an operating crew of nine men and costs some $8,000, has given way to the mammoth 25-cubic yard electric unit at Cananea, Mexico. Likewise, the 3-cubic yard, side dumping, hard rubber-tired World War I Liberty truck of the Jerome, Arizona pit has yielded to the modern 200-ton hauler costing in excess of one-half million dollars. Improved performance has accompanied these expansions in capacities, and despite higher investments and associated operating costs, the total expense of moving material has trended generally downward. This accomplishment stands as a tribute to the ingenuity of those associated with the open pit mining industry.

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XII. MATERIALS HANDLING RELATED TO MARINE BULK TERMINALS

A.D. Fernie*

Over the past decade there has been an increasing world-wide demand for bulk minerals in the primary and semi-finished states. Seaborne bulk tonnage has more than doubled as has the average size of the ships carrying commodities. To satisfy this dramatic growth in raw material requirements, mines, processing plants, and land transportation systems have increased in size and number accordingly, and to provide a connecting point between land and sea, the need has developed for large-bulk shipping and receiving terminals, with capacity to handle international ocean-oriented bulk traffic flowing steadily to a port in small parcels for dispatch in large tonnage lots, which upon arrival at their destination are rapidly discharged into a distribution system in small parcels or a continuous production plant. (See figure 88)

In this paper some of the more important parameters for the design and operation of storage and shipping facilities which play a key role in the step between producer and consumer will be examined in detail. Storage, reclaim and shiploading represent a significant part of the final cost of a commodity to the consumer, and careful examination of the materials handling concept as a part of a total transport system is therefore important. Planning should be thorough, as this is the basis for all further development, marketing, financing and operating at a profit.

Types of terminals

Apart from installations designed to handle grain and chemical liquids, there are two basic types of bulk storage and shipping facilities:

(a) Custom terminals. These are frequently designed and constructed to handle one commodity from one or two producers, and as such are designed around a known annual tonnage, often a defined or projected total tonnage. This type of terminal is perhaps the easiest to design because of the known parameters and is often owned and operated by a producer or subsidiary company;

(b) General bulk terminals. These are more difficult to design because of the multiplicity of commodities handled, the number of grades and producers of each commodity, the wide range of rolling stock at the receiving end and the variety of vessels at the outgoing end. In addition, there are problems of cross-contamination between products and scheduling of trains and ships; and storage capacity for the various products.

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* Materials handling specialist, Wright Engineers, Ltd., Vancouver, British Columbia, Canada.

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Large Storage/Reclaim System
(1,000,000 tons, minimum)

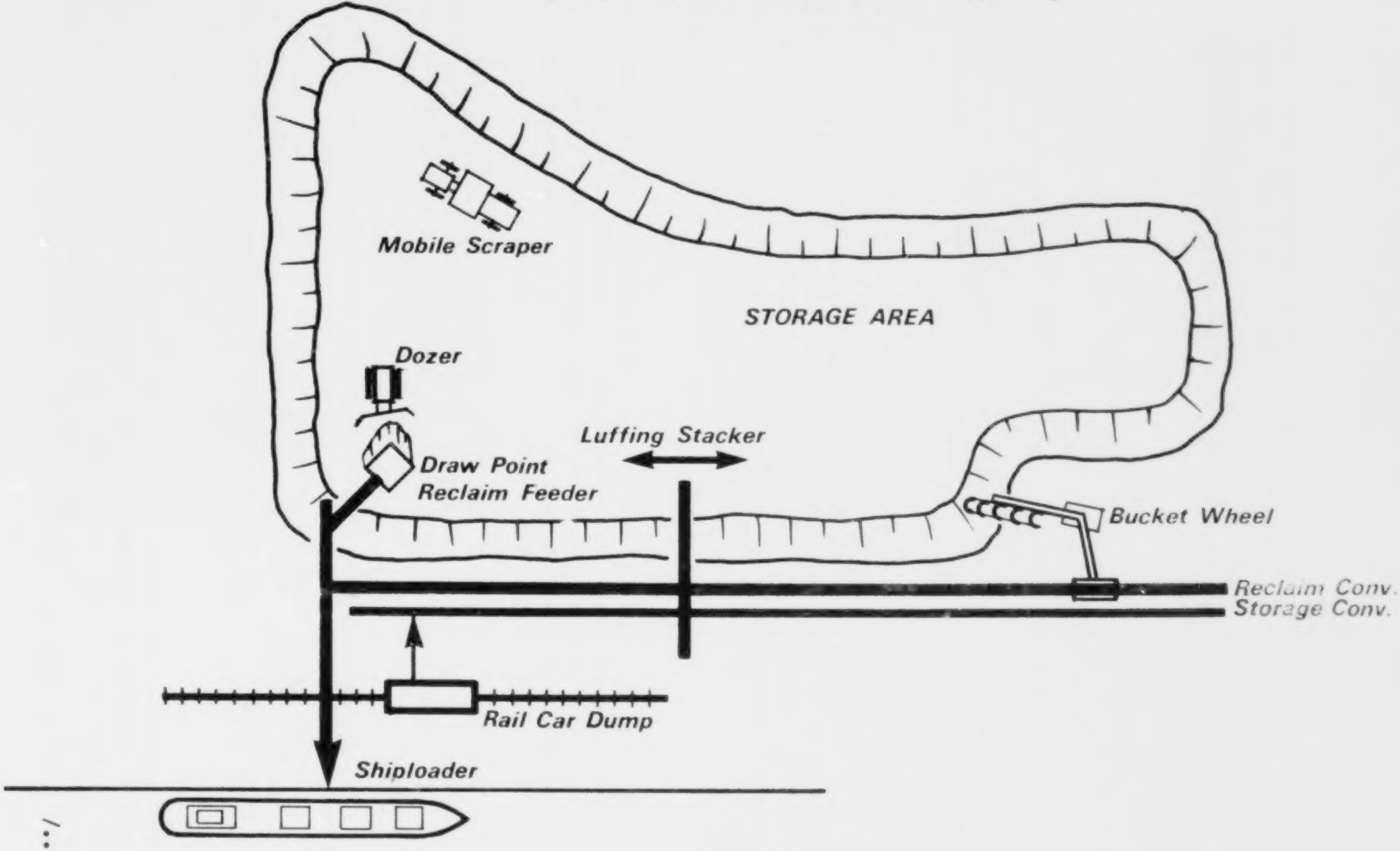
Mobile Scraper
STORAGE AREA
Dozer
Luffing Stacker
Draw Point
Reclaim Feeder
Bucket Wheel
Reclaim Conv.
Storage Conv.
Rail Car Dump
Shiploader

Figure 88

/...

Unlike the custom terminals, the general terminals must market their facility, and very often they take on short-term contracts as well as longer-term in order to keep as much tonnage flowing through as possible. Since tonnage on an annual and projected basis is much more difficult to ascertain, and shifts in tonnage requirements and commodities can occur on the world market, throughput at the general terminals varies substantially. This factor requires careful consideration in the design so that any major shift in commodity tonnage can be compensated by the increase of another commodity or perhaps even the introduction of a new customer whose commodity can be handled over the same facilities.

A terminal is not simply a collection of elements, such as a dumper, stacker and shiploader, but an integrated system. The terminal might be complex, depending on the number of interlocking elements, the number of options requiring flow control and the presence of isolated lines for commodity routing through the system.

Some important ingredients in terminal design are flexibility, adaptability and innovation. If these factors are ignored, with the accelerating rate of change, the time between initial studies and actual operation might, in some cases, render a facility inadequate or obsolete at startup. The risk factor in predicting commodity movements over the next decade is high and places an obligation on the planners and designers of systems to build into the concept, options to reduce the risk of obsolescence.

Chaguaya

Chaguaya is the world's highest bulk terminal installation located at 12,500 feet above sea level on the east shore of Lake Titicaca, Bolivia (figure 89). Completed in the spring of 1969, this terminal is part of a transportation system designed to move lead and zinc concentrates from the Matilde mine to the Pacific port of Matarani in Peru.

The terminal has a yearly throughput of about 100,000 metric tons of concentrate and consists of a dewatering system, a concentrate storage area, reclaim and weighing equipment, a transfer conveyor and a dock with a travelling-type shiploader. The concentrate enters the terminal in the form of a slurry, through a nine-mile long pipeline. It leaves the terminal as dewatered concentrate, loaded into gondola-type rail-cars on a self-propelled train ferry for the crossing of the lake and further haul down to the Peruvian coast. When the train ferry is in position along the dock, the dewatered concentrate is reclaimed from the storage area with a front-end loader and loaded into a steel hopper. A belt feeder, mounted under the hopper, transfers the concentrate on to the 1,425 foot long dock conveyor which in turn, feeds the shiploader.

The shiploader can travel a total of 157 feet along the dock, and its shuttle boom can be extended or retracted 43 feet across the width of the ferry. The average loading rate is 400 metric tons per hour. A 20-ton capacity stiff-leg crane, powered Uni-Flote rafts, and a transit warehouse are provided to handle incoming general cargo and supplies for the mine and community.

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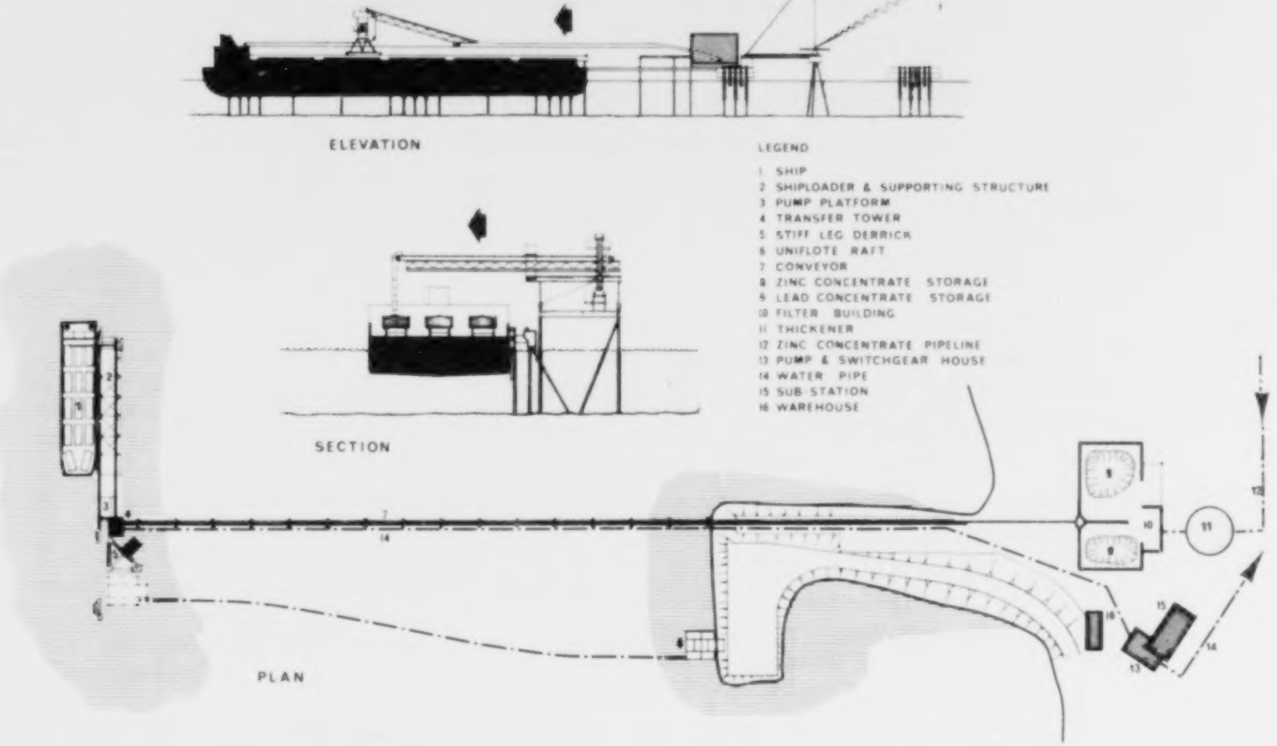
ELEVATION
SECTION
PLAN
LEGEND
1 SHIP
2 SHIPLOADER & SUPPORTING STRUCTURE
3 PUMP PLATFORM
4 TRANSFER TOWER
5 STIFF LEG DERRICK
6 UNIFLOTE RAFT
7 CONVEYOR
8 ZINC CONCENTRATE STORAGE
9 LEAD CONCENTRATE STORAGE
10 FILTER BUILDING
11 THICKENER
12 ZINC CONCENTRATE PIPELINE
13 PUMP & SWITCHGEAR HOUSE
14 WATER PIPE
15 SUB-STATION
16 WAREHOUSE

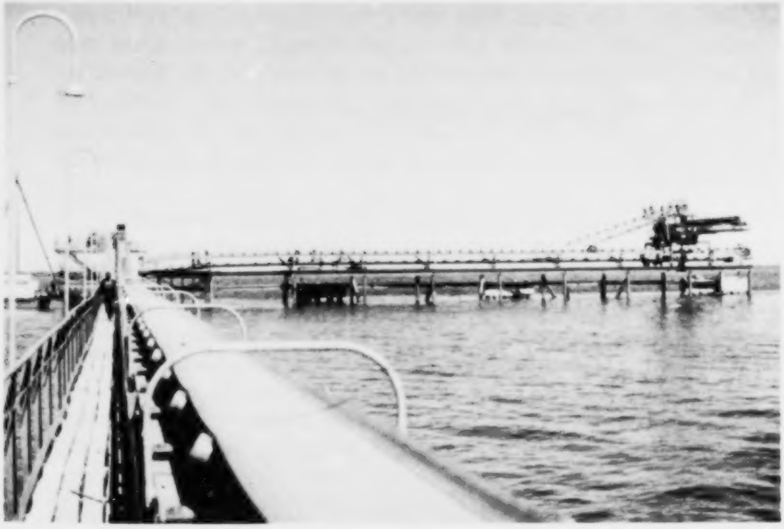

Figure 89

Causeway, Dock and Shiploader
PORT OF CHAGUAYA
Lake Titicaca, Bolivia

/...

B. Determining the terminal concept

A number of basic parameters should be defined for a bulk shipping terminal installation. These are:

1. Annual tons of commodity and total projected tonnage

2. Size of individual cargoes:

   To the terminal
   From the terminal

3. Mode of shipping:

   Receiving, that is, by rail-truck, pipeline or conveyor
   Shipping by barge, general cargo vessel, bulk carriers or slurry loading

4. Inventory at terminal:

   Value of product
   Deterioration, contamination
   Method and type of storage
   Number of grades, commodities
   Size and frequency of shipment

5. Commodity characteristics:

   Flow characteristics
   Specific gravity
   Moisture content
   Angle of repose
   Angle of withdrawal
   Bulk density
   Size analysis
   Degradation limits
   Special characteristics: corrosive; explosive risk; combustible (spontaneous)
   Dust-collection-suppression
   Freezing problems

6. Operation of the terminal:

   Private terminal: Can be operated by mining company personnel if the terminal is designed strictly as a bulk shipping facility handling the mining company's commodity only.

/...

Custom dock operation: Usually falls under the longshoremen's union and if the terminal is handling a very large tonnage a permanent crew can be assigned with special labour agreements drawn up to take care of receiving irregularities and shiploading times.

General bulk terminal: Usually falls under the longshoremen's union and generally operates under the normal union agreements.

C. Receiving

The reception of commodities at bulk terminals varies as the mode of transport of the commodity depends on the distance from producer to the bulk terminal, the tonnage of commodity to be moved on an annual basis and the type of commodity being transported.

Once the mode of transport has been selected, it becomes necessary to decide on the type of carrier best suited to the commodity, taking into consideration the following factors: (a) dust loss; (b) flow characteristics; (c) effect of weather and moisture addition, freezing, drying; and (d) value of commodity.

In order to adequately cover this subject, four types of operation will be considered:

1. A specific commodity terminal under 500,000 tons per year

Commodity can be received either by truck or rail for this size of movement. The most important factor to consider for this quantity is the capital investment required to unload the mode of transport efficiently. For instance, a good example of a proven method for unloading from gondola rail cars is by use of a Gradall mounted on a frame under which the rail-car can travel. This system requires a Gradall operator and four cleanup men and can discharge three cars per hour (see figure 90). The direct cost, labour and machine (owner/operator) is about $0.42/ton plus an annual write-off, interest and maintenance cost of $25,000. The capital expenditure for this system is less than $100,000, and although it is labour intensive and does not have a high unloading rate, it is economically ahead of a mechanized car dumper up to 800,000 tons/year.

Another method of unloading is the discharge of containers carried on rail-cars (figure 91). A system for lifting the containers from the cars, rotating them and replacing them on the cars requires a degree of sophistication but the equipment required is well within the economics for an operation with as little tonnage as 300,000 per year for a high value commodity being handled. For instance, it would not be economical to unload under a 1/2 million tons of coal using this system owing to the low market value of coal as compared to concentrate. This method is part of a total system concept and requires consideration on this basis.

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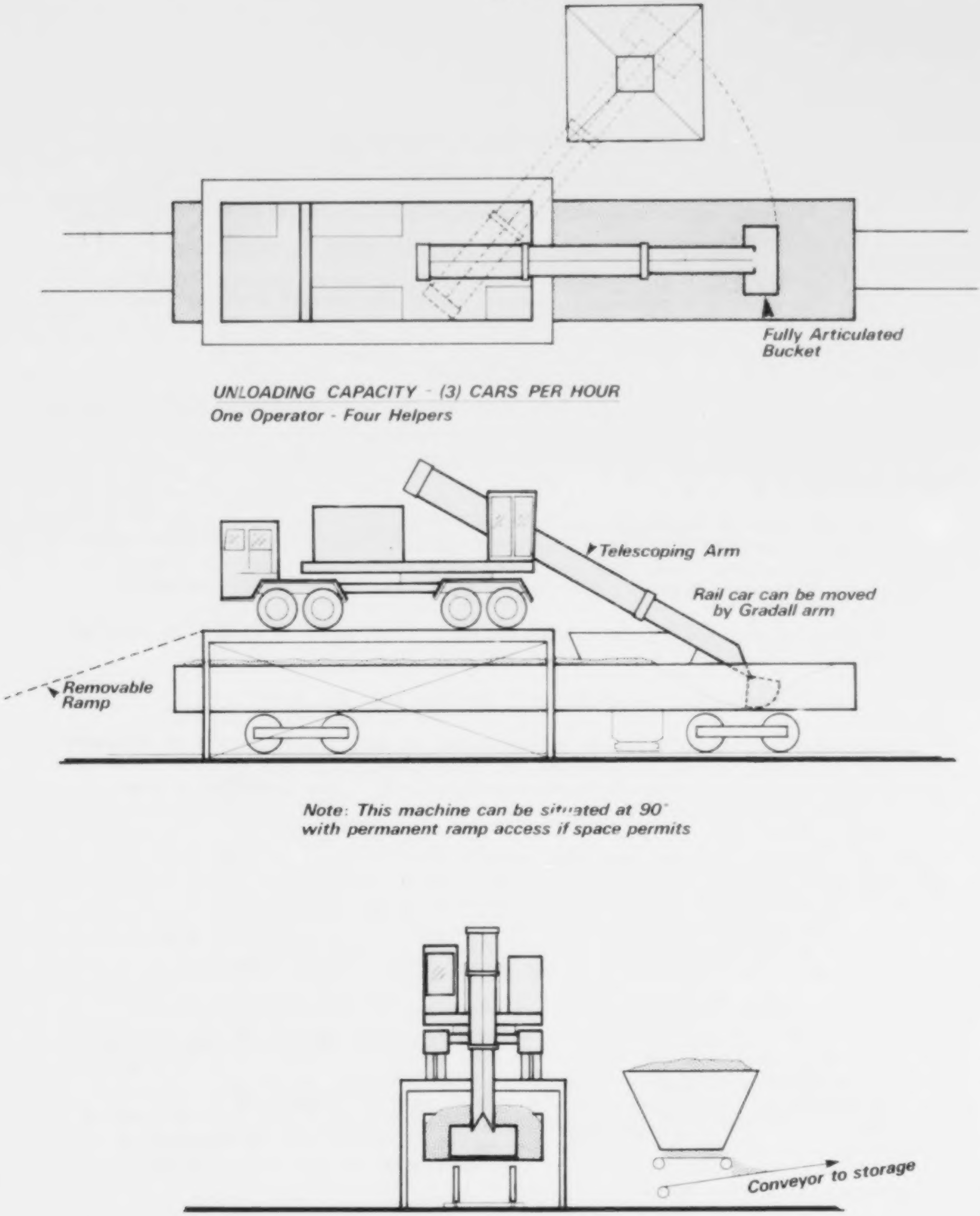

Fully Articulated Bucket
Telescoping Arm
Rail car can be moved by Gradall arm
Removable Ramp
Conveyor to storage

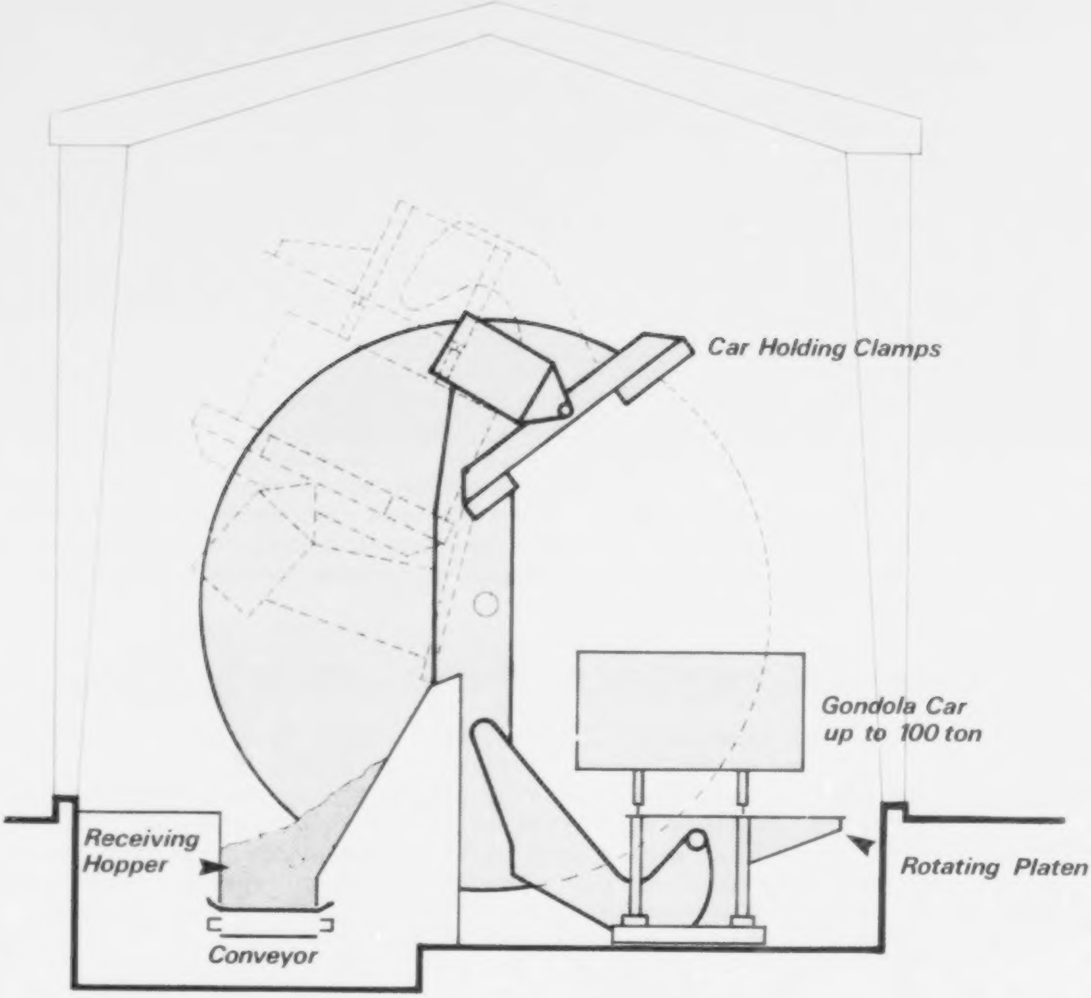
UNLOADING CAPACITY - (3) CARS PER HOUR
One Operator - Four Helpers

Note: This machine can be situated at 90°
with permanent ramp access if space permits

GRADALL BUCKET - RAIL CAR UNLOADING

/...

Figure 90


Car Holding Clamps
Gondola Car up to 100 ton
Receiving Hopper
Rotating Platen
Conveyor

SIDE DUMP RAIL-CAR UNLOADER

Figure 91

/...

Commodities which are free-flowing require a minimum type of installation in the form of a below-track hopper for single-hopper bottom or side discharge. Covered hopper cars of fertilizer, phosphate rock, dried concentrate and open-top hopper cars carrying coal and ore are suited to this receiving method.

Delivery of up to 300 tons/day by truck is often the case for small tonnage movement. Truck unloading methods differ depending on the type of box or container. Some of the units used are tanks with hopper bottom gates, box trailers into which a small front-end loader can bale the concentrate into a receiving hopper, side and end dump trucks and semi- and pup-trailers.

This method of transport is usually employed for tonnages up to 150,000 tons/year. Little capital expense is required to accommodate this system and with the small but steady shipments received, a front-end loader or elevated ramp system is more than adequate.

2. A specific commodity terminal for 600,000-1,000,000 tons/year

For this size of movement, more sophisticated equipment can be contemplated for unloading as the increased operating labour cost soon surpasses the operating and capital recovery cost required for a more mechanized system equipment. As an arbitrary breakpoint 800,000-1,000,000 tons/year is generally accepted as the average. For this size of movement, rail-cars are employed as the mode of transport except in unusual circumstances. The type of equipment used for unloading small rail-car lots in open-top gondolas could be a side-dump type rail-car unloader which has a capacity of up to 20 cars per hour rather than the more sophisticated high capacity rotary-type dumper. The side-type dump machine can be installed for under $700,000, including a device for removing rail-car covers and replacing them after discharge. This machine requires minimum subsurface foundations as compared to the rotary dumper.

3. A general bulk terminal for handling over 1,000,000 tons/year

A general bulk terminal receiving facility must be capable of accommodating many types and sizes of rail-cars. A rotary car dumper for discharge of open-top gondola cars and a bottom dump receiving hopper for covered or open hopper cars may be required to handle a wide range of commodities (figure 92). A review of the rolling stock that may be used should be undertaken, as many open-top hopper cars can be rotated, which is by far the least expensive way of discharging rail-cars. The direct operating cost is about $0.11/ton, plus an annual write-off, interest and maintenance cost of $190,000.

The rotary car dumper used for large tonnage and unit train movement can handle up to 40 cars per hour if rotating car couplers are used. If the train is broken and each car is spotted in the dumper, 25-30 cars per hour is the maximum capacity. The capital cost for a rotary car dumper installation is about $1,000,000.

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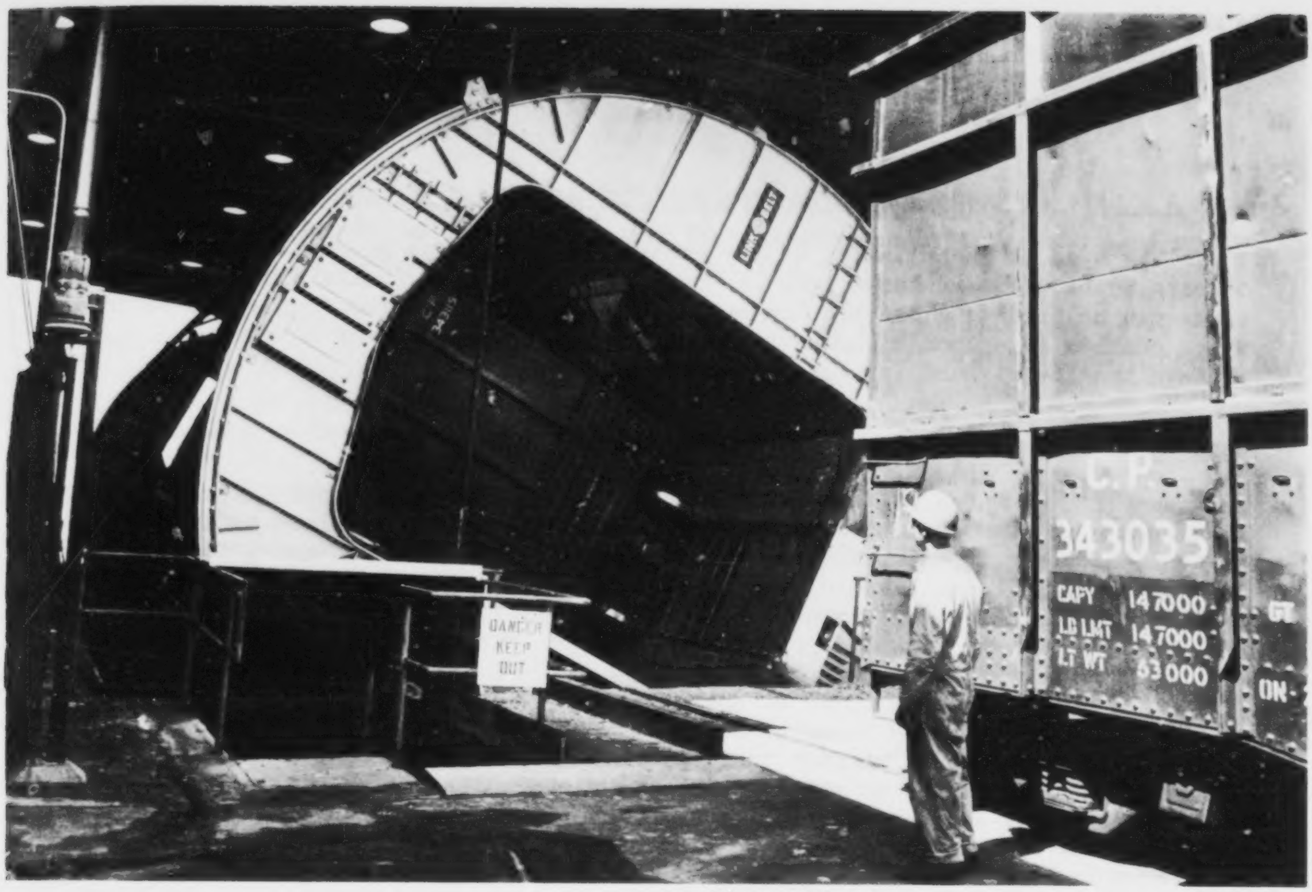
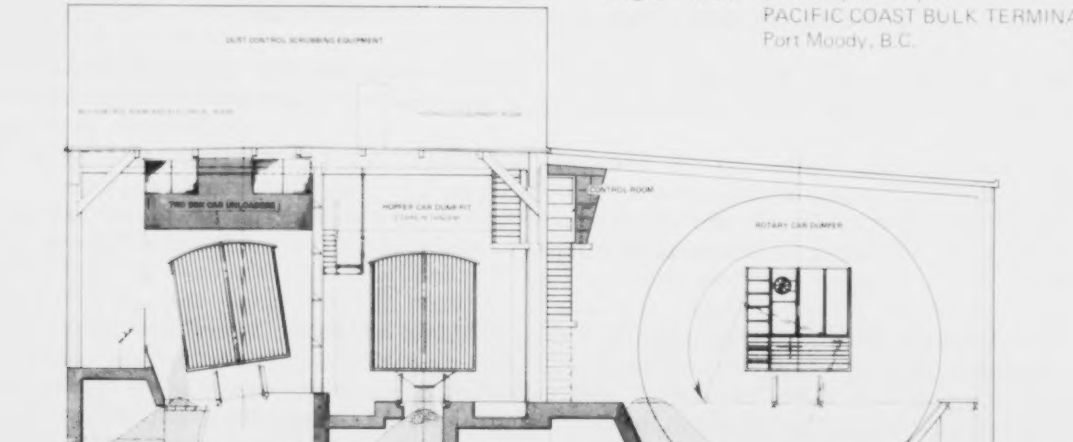

DANGER
KEEP
OUT
C.P.
343035
CAPY 147000
LD LMT 147000
LT WT 63000

Figure 92. Car Dumper Complex
PACIFIC COAST BULK TERMINALS LTD.
Port Moody, B.C.


DUST CONTROL SCRUBBING EQUIPMENT
TWO BOX CAR UNLOADERS
HOPPER CAR DUMP PIT
(2 CARS IN TANDEM)
CONTROL ROOM
ROTARY CAR DUMPER
VIBRATING FEEDERS
CONNECTING TUNNEL
BELT FEEDER

/...

4. A single or known commodity terminal for handling over 1,000,000 tons/year

This type of installation is the easiest to design, as generally the type of cars to be used are known long in advance. The size of trains and turn-round times can be predetermined and the annual tonnage and projected tonnage is defined, which enables an economic study to be carried out optimizing the design to achieve the lowest handling costs possible.

General

In all cases a receiving facility should include dust control equipment, and in the case of a general terminal, cross contamination between products is to be avoided by careful planning and design, allowing for washdown and cleanup. In some cases rail-car cleaning may be required if the cars are to be loaded with a different commodity for backhaul.

D. Stockpiling

A well planned bulk storage facility should efficiently handle the intermodel transport of commodities by streamlining the flow, which differs in time and volume on arrival and departure. The case for stockpiling or surge capacity is based on this requirement and can be attributed to two main factors:

(a) The continuous receipt of a commodity on a small unit basis to be accumulated for shipment in large export tonnage on-board ship;

(b) The necessity for surge storage to take care of rail and ship delays, labour problems at the point of origin, terminal, or foreign receiving docks.

In some special cases, it is possible to avoid storage entirely and ship direct from land transportation over a shiploader. A careful study must be made to determine the demurrage charges for rail-cars and ships as it is rarely possible to avoid delay. Scheduling ships to coincide with trains is extremely difficult.

When considering storage areas and stockpiling methods, there are many factors which require careful analysis and investigation. The following sections describe some of the more important steps in the process of evaluation but by no means cover all the possible combinations and alternatives. This paper does not deal with blending in the stockpile as this is rarely a requirement of a bulk shipping terminal.

1. Basic parameters

(a) Stockpile size. The size of storage required can be evaluated on the basis of an empirical number such as 10 per cent of the annual throughput, which is a generally accepted starting point. Another method is to determine the size of the shipments and probable frequency of ships relative to transportation and ships delays, inventory of product at the point of origin and general over-all scheduling difficulties which can result from weather conditions both land and sea, seasonal shipping or production fluctuations;

(b) Stockpiling rate. The rate of stockpiling will depend largely on the receiving method but it can be generally said that in most cases the machinery, whether it be fixed or mobile, should have excessive capacity over and above the discharge rate of receiving equipment so that it does not become a bottle-neck in the system;

(c) Stockpile height. The height of a stockpile is governed by ground support limitation, possible material degradation, practical building height, reclaim method and consideration of environmental restraints;

(d) Real estate. Waterfront areas especially in the region of a developed harbour are becoming prime real estate properties. Not only is the price of land or lease agreement becoming costly, but assessments and taxes are increasing at a high rate per year. Generally speaking, a bulk terminal requires a large plot of land which should be oriented away from populated or general industrial areas and have the potential for expansion. The concept of inland storage on land of lower value with conveyor or pipeline transport to dockside deserves consideration;

(e) Multiple products. The size of the storage area and configuration required, the shape of the pile(s) and the possible problems of cross contamination, if more than one commodity is being stored, must be considered. It is also necessary to determine if there are various grades, types or producers of the same product requiring separate storage areas, or whether there are a number of producers of different commodities requiring storage with complete isolation for each commodity;

(f) Civil structural requirements. Covering the storage area will play a role in deciding on pile configuration and influence the materials-handling system. The problems of open storage-area drainage, drainage from storage buildings and access for snow removal and cleanup must be carefully considered. The base for a stockpile will depend on reclaim requirements. In some instances, total reclaim is not required enabling the storage area to have a commodity base. If total reclaim is required, the base material will have to be determined. A concrete floor is considered to be the best because it can carry the weight of reclaim machinery if required, and can be made to resist deterioration caused by the product being stored;

(g) Climatic conditions. Freezing conditions may occur in a pile causing reclaim difficulties. Dust-loss or contamination from wind can occur and create severe problems. Snow and rain can affect open storage to the point of unacceptable moisture addition or damage to the product.

/...

2. Stockpile configuration

(a) Conical pile (figure 93). The simplest stockpile is a single cone formed by a single fixed point discharge of a conveyor. The conveyor can be equipped with a telescoping spout which will assist in controlling dust as the material is deposited on the cone. It is certainly the most economical shape of pile, is easily automated, and can be readily enclosed.

The cone is generally limited in capacity and has the distinct disadvantage of promoting segregation of product within the pile which may or may not be a problem. Very often large quantities of commodity are required for long-term storage, for instance, over a winter season. In this case a cone pile can be spread by bulldozers or earth movers to create a very large storage pile. This method of storing material on a large-scale basis requires little capital investment and is very efficient without carrying too large an operating cost;

(b) Multicone pile (figure 93). A modified version of the cone pile is to distribute a fixed point discharge by reversible belt to create two cone piles, or by shuttle belt to build multiple cone piles. This method is often used where several grades require storage. It is limited in size and has the problem of segregation;

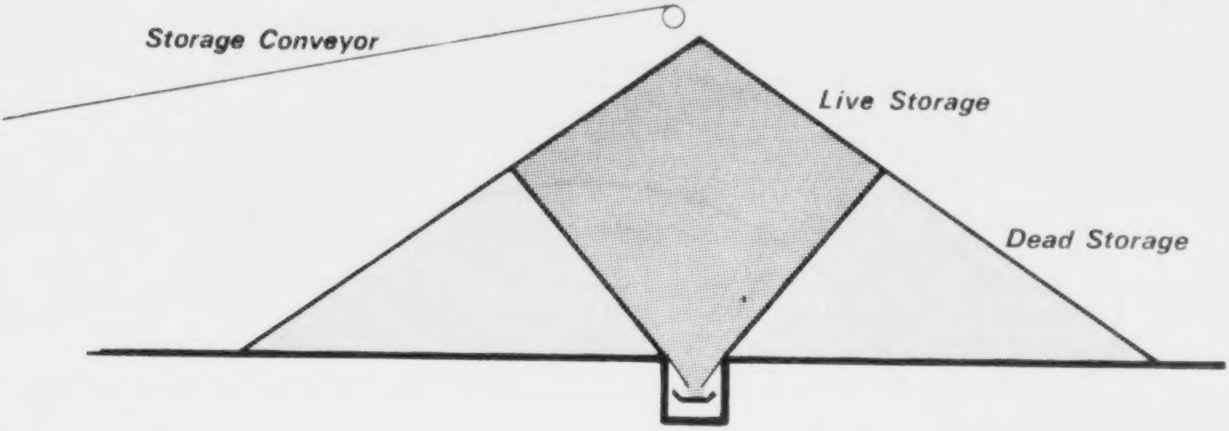
(c) Kidney pile (figure 94). The kidney pile is created by a radial stacker which comprises a fixed or luffing boom rotating about a central point to form a semi-circular pile. The fixed radial stacker boom can be equipped with a telescoping trunk to reduce dust when depositing product on the pile. The most efficient method of stockpiling with a machine of this type is by using a luffing boom which not only controls dust but reduces the fall of material onto the pile which is important when degradation must be avoided.

The angle through which rotation takes place is generally about 120 degrees in order to assist in straight-line tunnel reclaim if the gravity method is used. If reclaim from below the pile is not a consideration, the radial stacker can travel through 260 degrees without any difficulty. Like the cone pile, material segregation will occur but to some degree, can be controlled with a luffing boom;

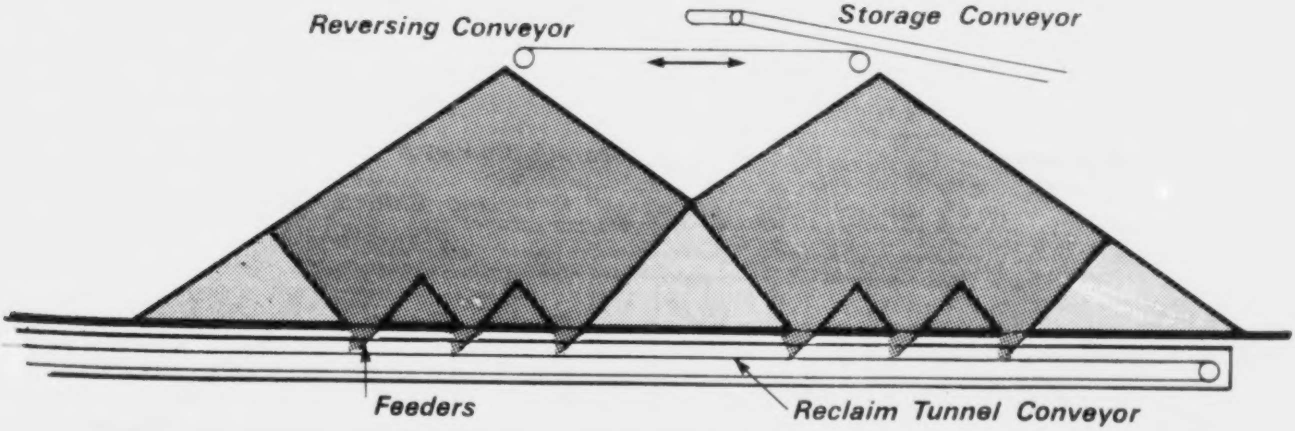
(d) Rectangular tent-shape pile (figure 95). The rectangular tent-shaped pile can be formed by either a conveyor tripper travelling over top on a supporting structure or within a building, or by a travelling stacker conveyor which transverses parallel to and at one side of the pile. This unit can be equipped with a fixed boom and telescopic chute for control of dust or with a luffing boom.

Storage capacity with this method of stacking is limited only to the height and length of the pile. It offers the advantage of expansion and the stacker can be remotely operated or automated. This method of stockpiling also allows the deposition of multiple grades or products in one length of pile, or the stacker can be made into a double wing-unit which can create stockpiles on either side of the runway track. This offers twice the stockpile capacity with a relatively small addition in cost of the stacker;

SIMPLE STORAGE PILES


Storage Conveyor
Live Storage
Dead Storage

Cone


Reversing Conveyor
Storage Conveyor
Feeders
Reclaim Tunnel Conveyor

Multicone

Figure 93

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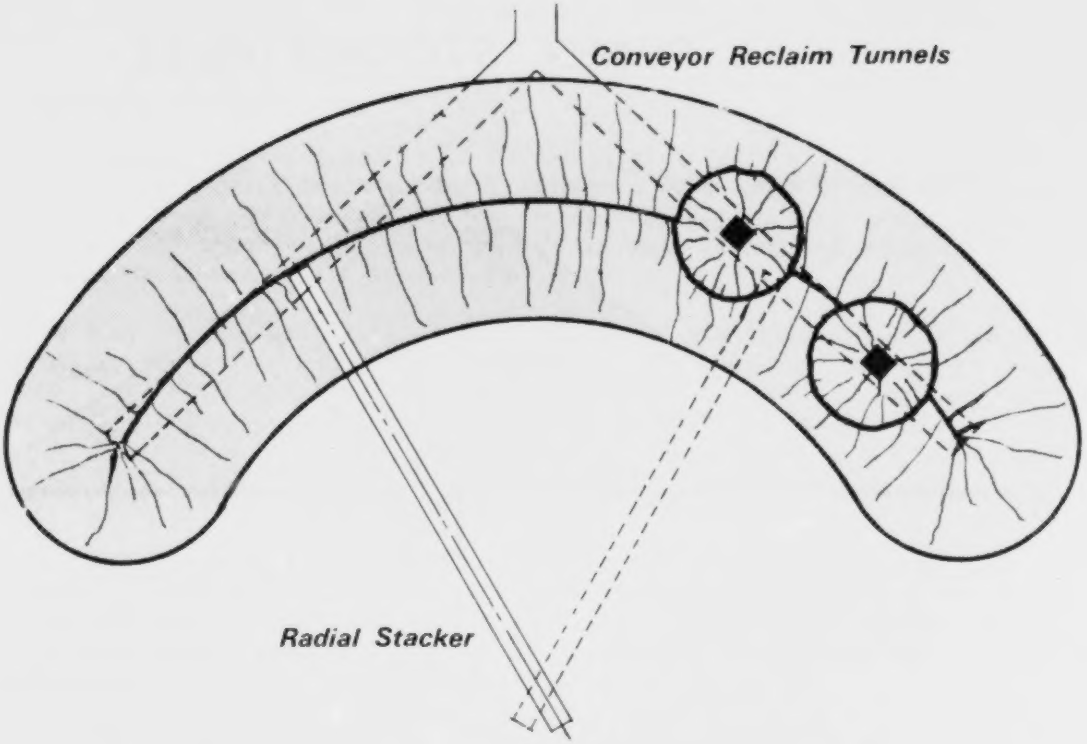
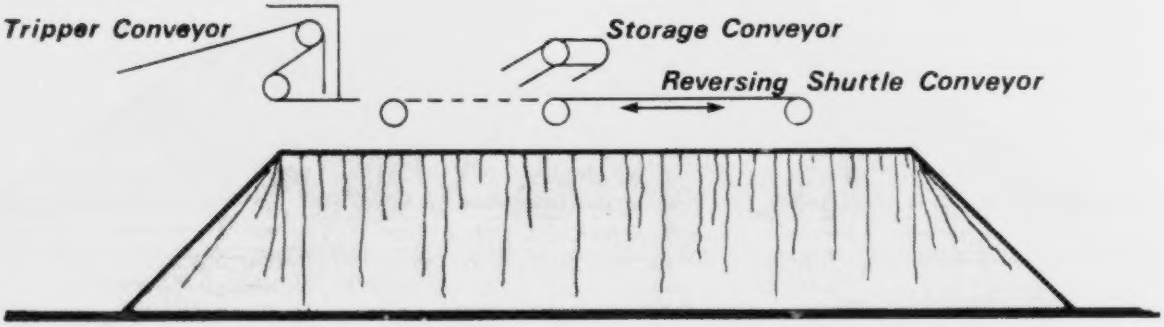

Conveyor Reclaim Tunnels
Radial Stacker

Kidney Pile

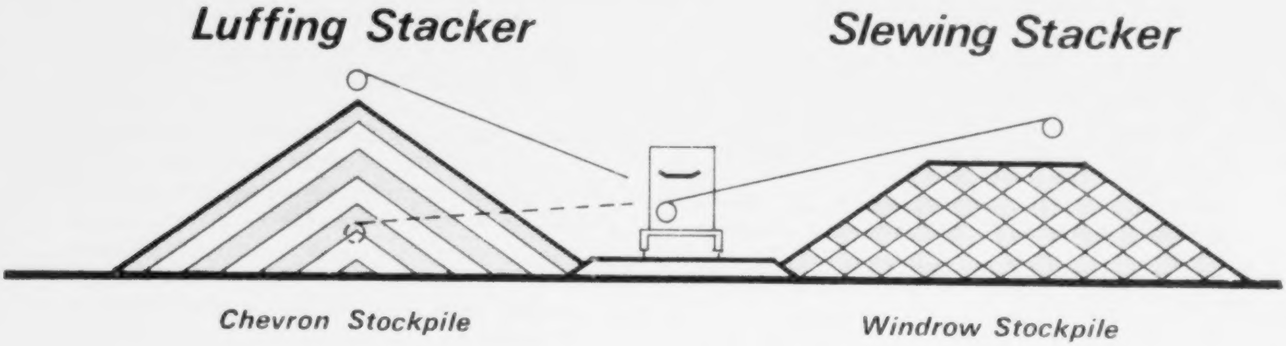

Tripper Conveyor
Storage Conveyor
Reversing Shuttle Conveyor

Rectangular Tent-Shaped Pile

Figure 94

/...

TENT & FLAT-TOP RECTANGULAR PILES


Luffing Stacker
Slewing Stacker
Chevron Stockpile
Windrow Stockpile

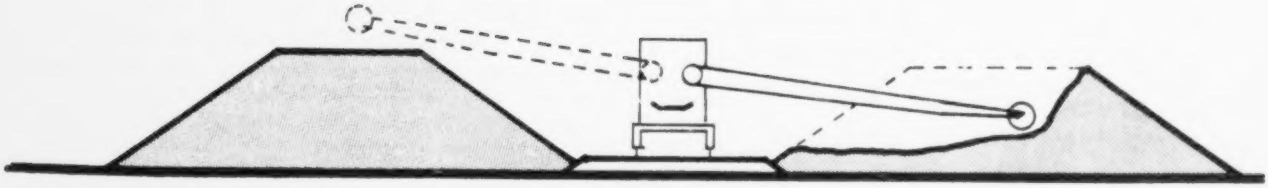

Stacker/Reclaimer

Figure 95

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(e) Flat-top rectangular pile (figure 95). A flat-top rectangular pile can be created by a further addition to the travelling stacker by equipping it with a slewing boom. The slewing boom can deposit material on either side of the stacker travel path, creating two piles and by positioning the boom at less than 90 degrees to the travel, can create a flat-top pile. With this machine, two methods of deposition in the pile can be employed. Using the windrow method, segregation of material can be controlled and at the same time will permit some blending. A very wide flat-top pile can be created by a travelling bridge carrying a conveyor and tripper for transverse material deposition;

(f) Slurry storage. In recent years slurry pipeline transport of solids on a large-scale basis has become a reality. This method of transport has now carried over to ships which can accept slurry loading. Before too long a number of bulk terminals may require storage for incoming slurry pipelines.

For a number of years Oregon Steel, near Portland, Oregon, has been receiving slurry from Marcona ships equipped with MarconaJets and ship-pumping stations. The slurried iron is received by a cutter-head suction dredge for distribution in the storage pond and is later reclaimed to the steel plant by the same dredge. The slurry storage ponds can be located a distance away from the prime waterfront area which can be a significant advantage in a crowded, industrially developed harbour.

Slurry storage ponds are now being constructed in a new terminal in Japan for receiving iron ore from Peru and New Zealand.

3. Commodity characteristics

(a) Flow characteristics. The flow characteristics of the commodity should be determined with particular reference to the angle of repose which can be expected under varying moisture control and particle size;

(b) Weight of material. The specific gravity of the material should be known, but even more critical is the bulk density. Bulk density can vary as much as 100 per cent between loose bulk or aerated to densely compacted material. This figure becomes critical in determining stockpile size as the bulk density on the stacker conveyors will be different than that of the material in the pile;

(c) Pile formation. Before a stockpile shape or method of deposition is selected, it should be determined if size segregation of the commodity will present a problem. If segregation is a problem, the flat-top rectangular pile should be selected using a luffing, slewing stacker which can deposit the material by the windrow method.

4. Economics

Unless there are some specific requirements related to the commodity which dictates the method of stockpiling, the following guidelines will assist in defining a stockpiling method:

/...

(a) Mobile equipment method. The most inexpensive way to build a stockpile is by dumping directly on the pile by end-dump truck. If delivery to the terminal is not by truck, the second, most inexpensive way is to use a front-end loader which can successfully build a storage area to a height of 40 feet by ramping up the pile. Both the above methods are considered for low tonnage operations only and reflect a small capital investment with a high operating cost;

(b) Mechanized conveyor method. Conveyor overhead trippers, or radial and luffing stackers are generally considered in the 500,000-5,000,000 tons/year terminal size and should be kept as unsophisticated as possible and still cover all the requirements for effective stockpile building;

(c) Highly mechanized conveyor method. For very large tonnages of a commodity, a luffing, slewing stacker concept provides an economical method of stockpiling. If desired, this unit can be automated, although it becomes somewhat complicated, and in most cases the degree of automation in a bulk handling terminal should be kept as straightforward and simple as possible. The personnel usually employed for this type of operation are not accustomed to sophisticated process plant type automated equipment.

E. Reclaiming

An effective and efficient reclaim system is without question one of the most difficult areas of concept design in a bulk terminal. Two basic and very often most lightly considered factors in designing a reclaim system are:

(a) To collect as much background information regarding the commodity as possible. If full information is not available regarding the handling characteristics, testwork should be carried out to determine characteristics, such as bulk density under different conditions and angle of repose and withdrawal;

(b) Ensure that when selecting reclaim equipment, a full appreciation between rated and average capacity is applied to the system.

Reclaim systems are a costly part of a bulk terminal and therefore deserve a separate economic analysis. By nature of the operation, a site is generally located adjacent to a body of water and the backup land is usually at an elevation close to water level. Ground conditions often present difficulties in providing adequate foundations, and ground-water problems make reclaim tunnels and sublevel installation quite costly.

Reclaim equipment generally must have excessive rated capacity over the system due to the variations in reclaim rate which are discussed in detail under each method. The following sections describe factors which are required for concept planning and an evaluation of reclaim methods.

1. Basic parameters

(a) Reclaiming rate. The rate of reclaim from storage is generally dictated by the shipping rate which is discussed later. When considering reclaim rate

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relative to shipping rate, it is most important to ensure that the rates discussed are defined, that is, design rate or average rate. When the average rate has been defined, it is most important to carefully examine the reclaim method selected and calculate an achievable rate, taking into consideration the various factors outlined in this paper for each method.

Many terminals have never been able to achieve their loadout design rate because of the lack of attention given during concept planning and design to achievable reclaim rates. The operation of reclaim areas is not fully understood by many technical people, including equipment manufacturers. Without actual operating data which are very difficult to obtain, many design concepts fall short of their goal;

(b) Reclaiming method. Selection of the reclaiming method will depend to a large degree on soils conditions and the water table, which will influence the decision on tunnels versus surface-type reclaim equipment;

(c) Reclaim demand. In some cases it may be necessary or advantageous to reclaim two types or grades of the same commodity simultaneously, which could influence the selection of a reclaim system. In the case of a general bulk terminal, it may be a requirement to reclaim two different commodities simultaneously and keep the products totally isolated for loading in different holds.

2. Material characteristics

(a) Flow characteristics. It is necessary to determine whether the material is free-flowing under all or some conditions, and to determine the angle of withdrawal. In the case of confined flow, it is advantageous to know the cohesive strength and sheer values which can be assigned to the material to properly size openings for feeders and hoppers.

If gravity reclaim is possible, it then becomes a question of determining the percentage of live material in storage and determining the cost of reclaiming the dead storage if total reclaim is required.

(b) Weight of material. It is very important to determine by testwork the bulk density of the commodity in, for instance, a front-end loader bucket or wheel bucket as compared to the density in the storage pile. Many commodities tend to "bulk" when they are excavated and the tendency is for their volume to increase in some cases by 20-25 per cent and their weight per cubic foot drop accordingly.

3. Reclaim methods

(a) Gravity reclaim. Tunnels beneath a storage pile with gates or feeder arrangements are a widely used method of stockpile reclaim. The storage of bulk material in bins and silos is a separate subject and is not dealt with in this paper.

/...

Two methods of tunnel reclaim are widely accepted: the first, consisting of feed points located in optimum positions below the pile is the least expensive of the tunnel reclaim methods. Unless the commodity is extremely free flowing, the live capacity will be about 40 per cent, the balance having to be mechanically moved to the feeder openings. With this method of reclaim, some blending from the point of view of size segregation can take place, but is not particularly applicable in a high capacity bulk shipping operation.

The second method, the slotted-type tunnel with rotary plow feeder(s) (figure 96) can reclaim 50-55 per cent of the material live and can provide a high tonnage rate reclaim over a very long pile. The capital cost of the tunnel is high, but if the stockpile is very large, and a large number of feeder reclaim points would be otherwise required, a rotary plow provides a viable solution.

(b) Mobile equipment method. The wheeled front-end loader and wheeled or track bulldozer equipped with a special "U" blade, reclaiming to conveyor loading pockets is a proven and reliable method of physically reclaiming material from storage. The capital cost for equipment is not excessive. The operating cost is high and for the number of tons moved, it is operator intensive. This type of reclaim equipment in a general bulk terminal handling up to 5 million tons/year has the advantage of being flexible to move about the terminal and reclaim various products.

When selecting this type of equipment for a reclaim system, the calculation of cycle times and actual operating time over a shift should be carefully evaluated. For example, in many operations seven hours of an eight-hour shift are worked. In addition the cycle times are based on a 50-minute hour. It also must be remembered that this type of operation is continuous cycling and is totally dependent on the capability and skill of the operators. It is therefore necessary to evaluate the class of operator which is available and determine what the low average cycle time would be.

In deciding on the type of mobile equipment the practicality of running a machine on the pile for reclaim should be considered. Many materials hang up and then cave, endangering the operator and the machine. When selecting front end loaders it is vital to size the bucket relative to the machine capability. In some cases where a heavy material is being handled too large a bucket is fitted to the machine which causes, on high cycling duty, very short transmission and brake life, both of which are costly maintenance items. The reverse is often true. A bucket on a front end loader is sized on tipping load on rough ground. A reclaim system does not usually require the loader bucket to be lifted to the tipping position and the machine is operating on a smooth surface. For these reasons the bucket capacity can often be 25 per cent greater than rate for the machine providing the bulk density of the product being handled is suitable.

(c) Mechanical handling. Three methods are described below: the bridge clamshell reclaimer, the bucket wheel, and the portal luffing scraper.

Bridge clamshell reclaimer. This method of reclaim requires a large capital investment and does not have the capacity of other systems of comparable cost.

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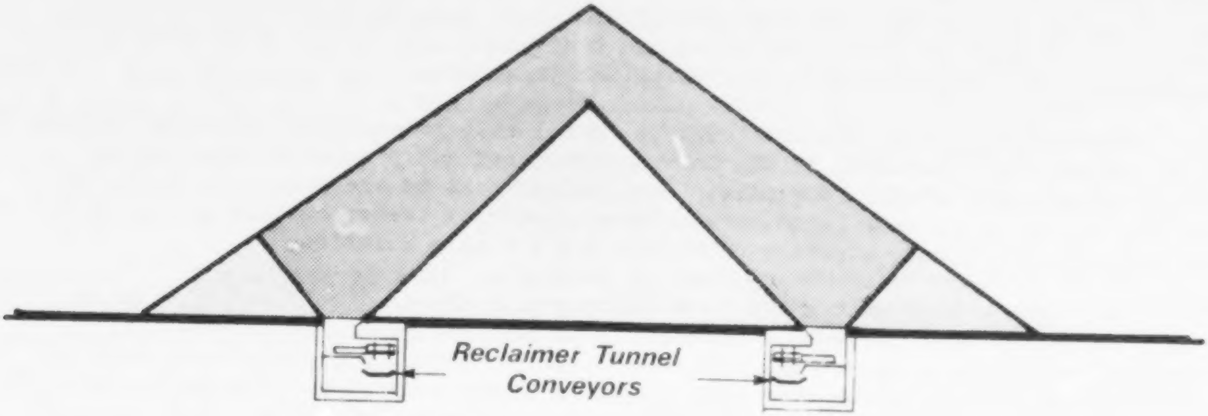
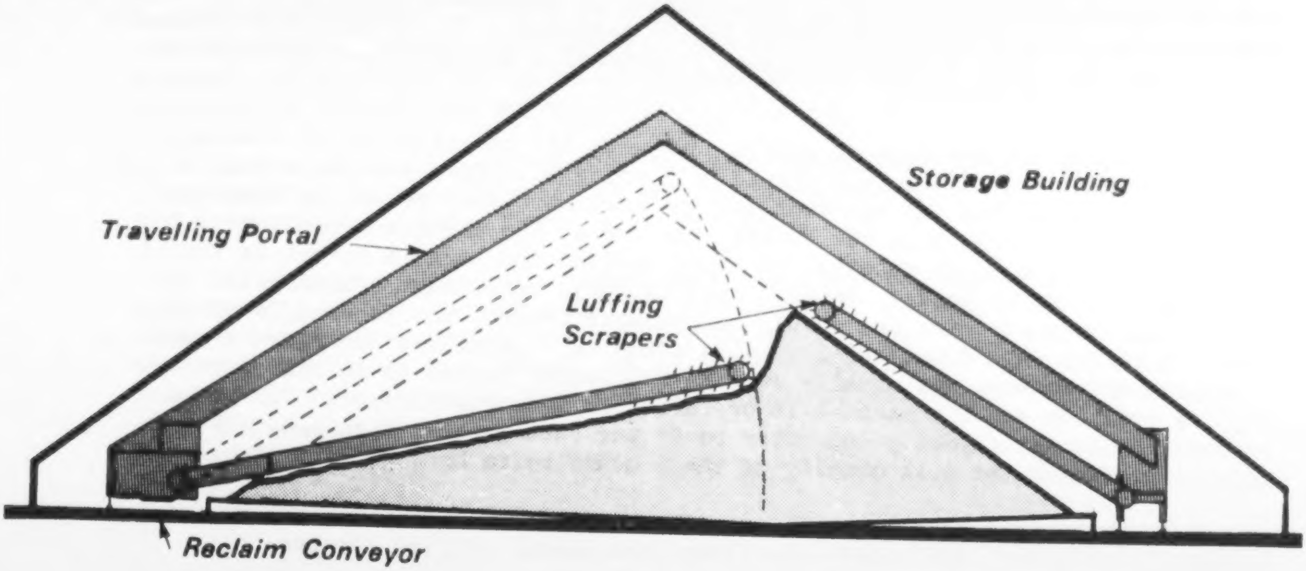
STOCKPILE RECLAIM


Reclaimer Tunnel
Conveyors

Rotary Plows


Storage Building
Travelling Portal
Luffing
Scrapers
Reclaim Conveyor

Portal Scraper

Figure 96

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It does however, have the distinct advantage of being able to move hard-to-handle materials which cannot be reclaimed by other means. It is also flexible, as it can select a digging position over a wide-long pile.

Bucket wheel. There are many designs of bucket wheels for various heights and shapes of stockpile. Bucket wheels are suitable for reclaiming relatively free flowing material and are most efficient for use on long rectangular pile configurations. There are two basic types of bucket wheels:

(i) Crawler-mounted. There are two types of crawler-mounted machines. One is a side mounted wheel which is manufactured in North America. The other type has a wheel mounted on the end of a conveyor boom with a tail boom conveyor for connexion to the main system. Some of the advantages of the crawler-mounted bucket wheel are they can reclaim 100 per cent of the stored material, discharge their load to either side of the pile and not being mounted on a runway can move to different piles for reclaim. The disadvantages for storage pile reclaim are significant and should be carefully considered before this type of machine is selected. The crawler machine generally has a short boom, therefore limiting the height of pile that it can effectively reclaim. If the material is very free-flowing there is no problem, but should it have a tendency to pack and hang up or undercut, then a serious drop in capacity will occur as the bucket will be subjected to slides and burying. The crawler wheel also has the disadvantage of having two conveyors, one of which has to co-ordinate with a transfer car on the main belt conveying system. When digging at high capacities and advancing at regular intervals it is often difficult to maintain this connexion without spillage and delay time in repositioning. The crawler-mounted machine also requires an operator and an attendant for either fueling if it is a self-powered unit, or for trailing cable and interconnexion observations.

(ii) Rail-mounted. The rail-mounted bucket wheel has fixed limits and a defined range providing a positive link to the main conveyor system. It can be automated but in a bulk terminal situation it is generally desirable to have an operator on the machine. There are two types of rail-mounted machines, one having the rails running down the centre of the stockpile which is the type most commonly used inside of storage buildings, the other having the rails mounted parallel to the long side of the storage pile.

The alongside storage pile rail-mounted reclaimer can change position rapidly and recover product on both sides of the rail, therefore serving two stockpiles. This type of machine is the most economical solution for a large storage area.

The rail-mounted bucket wheel leaves a quantity of dead storage which requires clean-up by front-end loader or bulldozer unless an embankment replaces the dead storage area along the far side of the pile.

It must be pointed out that the rated capacity of bucket wheel reclaimers and the rate actually achieved vary widely. When commencing and completing the reclaim of the pile cross section, the reclaim rate can be as low as 20 per cent of rated. Only in heavy digging in the body of the pile does the machine generally achieve its rated capacity.

When two machines are reclaiming to one conveyor, a surge bin should be considered to even out the flow to a value of about 50 per cent more than the average of one machine. It is unlikely that both machines will dig at maximum capacity for any extended period of time and therefore it is not prudent to supply double the rated capacity within the system.

(iii) Bridge-type bucket wheel. This machine is most suitable for reclaiming from a storage pile when blending is required. They are very expensive and maintenance costs are high. They are generally not required in a bulk terminal situation.

(iv) Combination stacker wheel reclaimer. This machine is very versatile and is applicable to many bulk terminals where stacking and reclaiming operations are not required simultaneously. When a commodity arrives at the terminal and is coincidental with a ship loading, the commodity can often bypass storage and be directed through the stacker/reclaimer to ship and in some cases combine with the reclaim operation. This equipment does provide an economical approach to storage and reclaim, as it performs two functions and can service two parallel piles. One disadvantage is the boom length can cover a much wider pile section for stockpiling than the wheel can reach for reclaiming. As a result the quantity of dead storage can be a significant percentage of the total. When operating this type of machine it is recommended to limit the boom outreach during normal operation to a distance to which the wheel can reach on reclaiming. If the storage pile becomes full, the boom can then be positioned for further outreach to provide additional storage capacity.

Portal luffing scraper (figure 96). The portal luffing scraper is an adaptation of the scraper chain continuous conveyor. The scraper chain has been mounted on luffing booms supported by a travelling portal frame. There are many types of carriers but the one most suited to bulk terminals is the portal frame.

This reclaimer is mainly applicable to inside storage, handling dry granular, free-flowing materials such as phosphate rock, potash and fertilizers. The maximum reclaim rate for one scraper is about 1,200 tons/hour and the machine is capable of producing a continuous flow of an average rate above 75 per cent. The commodity is scraped to one side of the storage building and deposited on a conveyor belt running parallel to the pile. The scraper boom travels along the pile, cutting a slice off the sloping face until it has cut from the angle of repose down to floor level.

The machine although limited in use is versatile and when properly applied provides a reliable reclaim system.

(d) Slurry reclaim. Slurried product may be reclaimed from holding ponds or bunker-type storage ponds.

Holding ponds. Slurry held in storage ponds usually arrives at the terminal in a pipeline. A floating cutter-head dredge is the most practical and economical method of reclaim and a system can be designed to handle up to 4,000 tons/hour.

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Bunker storage. Marcona Mining in Peru has 75,000-ton and a 150,000-ton bunker-type storage ponds. They are equipped with Marconajets for recovery of storage which is pumped in slurry form directly onto ships. The method of slurry recovery is through sumps located at each jet. The slurry is then directed from the below bunker sumps to a pumping station for transport. Basically this system of reclaim is identical to that installed on the Marcona ships described later in this paper.

F. Shiploading

In order to select the type of shiploader most suited to the terminal operation and at the same time satisfy the shipper's requirements, it is necessary to understand the factors which determine vessel turn-around time. Some of the seldom recognized circumstances which lead to a greatly reduced gross loading rate resulting in extended lapses of time are discussed and should be carefully considered for each case under study.

Although beyond the scope of this paper, determination of ship frequency and scheduling is probably the most difficult aspect of determining the number of berths, shiploaders and system capacity required. Ship frequency is affected by shipping companies' schedules, weather, strikes, breakdowns and unloading delays, sometimes resulting from unavailability of berthage at the discharge port.

Berth occupancy can be determined on a mathematical model simulating port behaviour. The queuing theory, analytical method and computer techniques can be applied depending on the complexity of the operation. No matter which method is selected for determining berth occupancy, turn-around time variables, such as tide, weather variations, delays and so forth, must be included (table 18).

The analytical method is generally quite satisfactory for single berth determination and can be used to approximate multiple berths. The queuing theory deals with events that are partly variable and partly random and involves the theory of probability.

From experience the number of variables is extensive and the type of terminal and the quantity and type of commodities being handled play a large part in determining berth occupancy.

In order to reduce the incidence of ship queuing, berth occupancy should not be over 50 per cent. In terms of shiploader capacity this means that the shiploader must be capable of handling the annual tonnage in half the available annual operating hours. In order to calculate the rated capacity of the shiploading system it is necessary to determine the average gross loading time, which might include hatch repositioning, ship service time, clearance to load, port warden clearance after loading, discharge of ballast, ship survey, tidal influence on ship-leaving time and pilotage scheduling. Other factors which can influence the loading time might be wave action or wind frequency.

Table 18. Determination of berth occupancy (example)

(Potash: 3 million tons/year; 300,000 ton storage capacity; 4,400 tons/hour loading system)

	Ship size (dwt)		
	10,000	30,000	50,000
Ships per year	300	100	60
Operating hours available, 3,400 (not including holidays, maintenance)	13.3	34	56.6
Shiploader rated capacity 4,400 (tons/hour)			
Average loading rate 3,300	3.0	9	15
2,200	4.6	13.7	23
Ships' service time (hrs)	24	18	12
Net time (High rate	27	27	27
(Low rate	28.6	31.7	35

The above example shows that for shipments of potash averaging 30,000 tons, berth occupancy is at the best 80 per cent, which is considered too high for this type of operation. From experience to date, and every indication is that the trend will continue, potash is shipped in small to medium-size bulk carriers. If this assumption is made, three alternate solutions to the above berth-occupancy problem are available: (a) to extend from a two-shift to a three-shift operation; (b) to install two shiploaders; or (c) to construct two berths serviced by one shiploader.

The chart also shows, in the case of the small vessels, an exceptional amount of service time. In practice the smaller ships require more service time and reduce the effectiveness of a high loading-rate.

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Table 19 shows some examples of rated capacity of a system, net loading rate and gross loading rate. These figures are taken from actual records of a number of general bulk terminals. These terminals are considered to be up to date and comparatively efficient. As can be seen by the examples on the table it is very necessary to analyse the situation carefully before setting the rated capacity of the shiploading system.

In some cases, shipping contracts have demurrage and dispatch clauses which provide some incentive for the terminal operator to achieve rapid turn-around time on the vessel, but in terms of the amount of dispatch money paid out in a year on these contracts the cost of providing a much larger loading rate is not justifiable. In some of the more recent bulk shipping contracts there has been no provision for demurrage and dispatch payment and extended delays have been taken care of in other ways.

In the case of large tonnage movements of a specific commodity, a shiploading system can achieve 75-80 per cent of rated capacity, provided it is backed up with a reclaim system delivering a continuous tonnage flow. Where many large systems fall short on loading capacity it is attributed to the inability to maintain a steady flow of material from storage.

1. Shiploader types

(a) Portable shiploaders. In some cases where a small tonnage of bulk commodity is being handled over a general cargo dock, portable conveyors, in conjunction with a portable shiploader, are employed. A system of this type has loading rate limitations because of size and weight restrictions but can be designed to handle between 500 and 1,000 tons/hour depending on the commodity.

The shiploader boom can be picked up by ships gear, cantilevered from a travelling frame or suspended from a dock crane.

This method of shiploading is usually limited to vessels under 25,000 dwt and results in a poor gross loading rate, as trimming, hatch moves and flow to the conveyor system reduce the average loading rate substantially.

Portable shiploaders are being successfully used in many port operations, such as Matarani and Callao in Peru, and Pacific Coast Terminals, Ltd. in Prince Rupert, British Columbia, Canada.

This system of shiploading results in minimum cost where a multi-use dock apron already exists and part cargoes of bulk commodity are normally shipped;

(b) Fixed luffing boom shiploader (figure 97). This type of loader is used for small movements of bulk cargo and is generally not acceptable for loading vessels of over 15,000 dwt. One of the main problems in having a fixed boom-length is trimming below decks, which with a fixed point discharge is time-consuming and in some cases causes difficulties in "topping up" a hold. A modified version of

Table 19. Examples of actual loading times

Commodity	Tons loaded	System rated capacity (tons/year)	Net rate	Gross rate
Zinc concentrate	23.866	1 500	1 110	837
Copper concentrate[a]	29.857	650	237	-
Lead concentrate	23.055	1 500	904	636
Lead and zinc concentrate (50 per cent each)	26.326	1 500	1 183	829
Sulphur	39.000	2 000	1 640	920
	24.000	2 000	1 410	330
	8.500	2 000	740	298
Coal	57.000	1 500	1 210	1 080
	61.000	1 500	1 040	635
	11.200	1 500	1 400	310
Potash	13.785	1 500	1 280	670
	31.775	1 500	640	400
	6.530	1 500	510	170

Note. The table clearly shows the wide range of net and gross loading rates that can in practice substantially derate a terminal's published capacity. The rates shown are low and high averages and are considered to be typical of a general bulk terminal operation.

[a] Total tonnage represents 10-part cargoes loaded onto difficult-to-trim vessels. Loading 'tween deck and underneath deckhead wings is time-consuming, making net loading-rate low.

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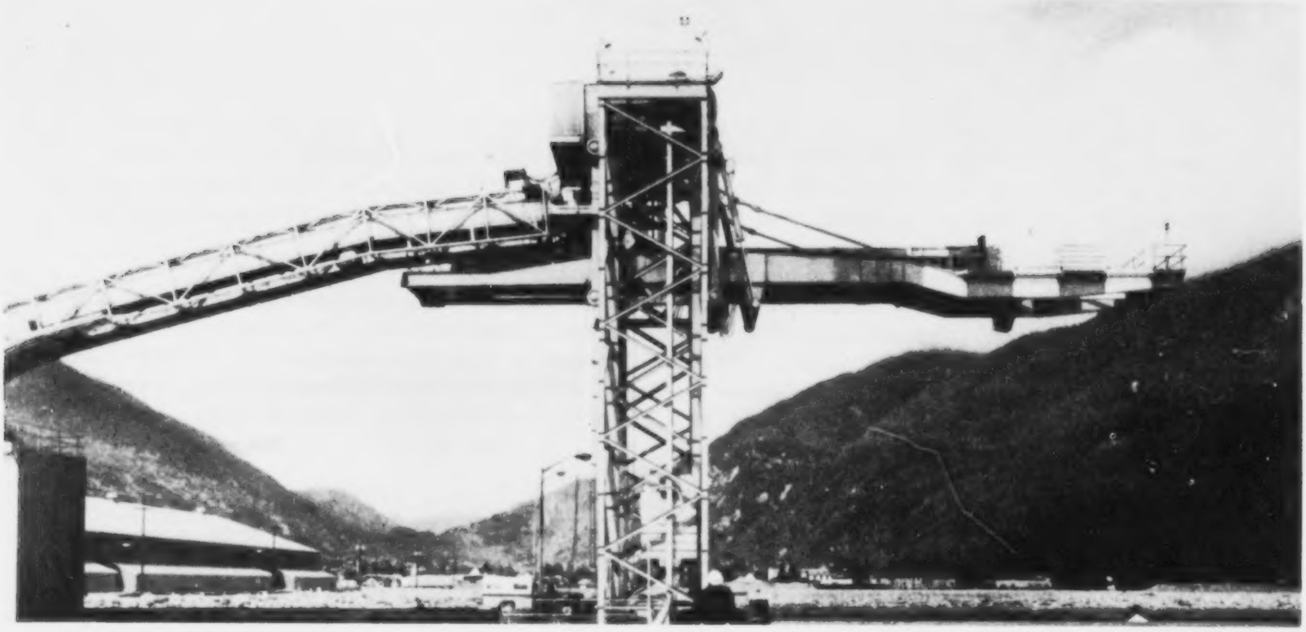
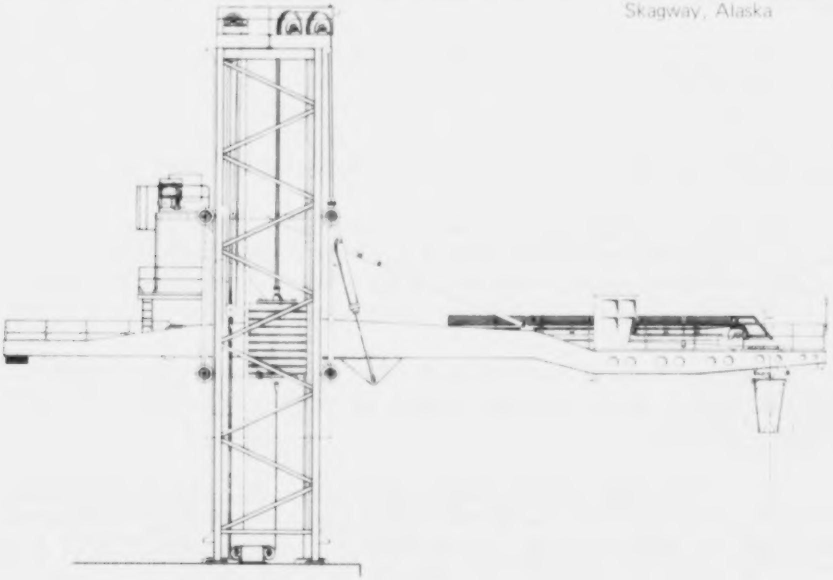


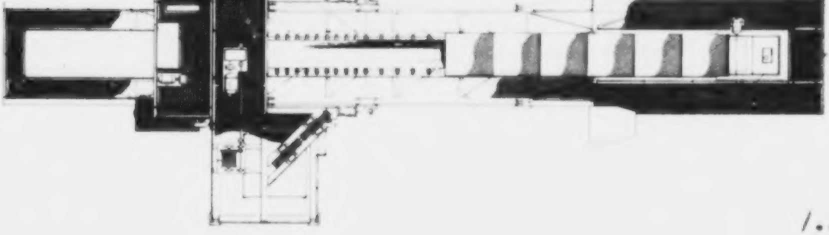
Figure 97. Fixed buffing loader

SKAGWAY ORE TERMINAL
White Pass and Yukon Route
Skagway, Alaska

ELEVATION



PLAN



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this type of shiploader provides a shuttle conveyor with the ability to move thwart-ships through a range of about 20 feet. This modification to the fixed luffing boom will allow loading of ships up to 35,000 dwt, and with a mechanical trimmer at the end of a spout can be relied on to trim a hold efficiently.

This type of loader is generally satisfactory for shipping up to 500,000 tons/year. Providing the dock and facilities are adequately designed for ship warping, the turn-around time of the vessel need not be extended much over that of a travelling shiploader. Table 20 shows a typical example of a travelling shiploader versus a fixed shiploader. The figures are taken from actual operating and statistical data. Both systems are being supplied from storage using the same reclaim method;

(c) Parallel travelling luffing shuttle shiploader (figure 98). This type of shiploader is particularly suited to handling annual tonnage of materials up to 3 million tons/year, and can efficiently load vessels with above deck cargo gear. Smaller vessels are often used for carrying up to 10,000 ton parcels or part cargoes of even much smaller quantities. This type of loader is versatile and should be designed for rapid traversing along the dock, as the number of hatch moves required for small vessels can cause a significant decrease in over-all loading rate

(d) Quadrant shiploader. The quadrant shiploader has a bridge which moves radially about a fixed point on the shore side and has a shuttle conveyor mounted on the bridge. This unit is not particularly suitable for loading smaller vessels, as reach to the various holds is effected by changing the angle of the boom through a radius which often interferes with deck gear or superstructure.

A single quadrant loader is seldom suitable for loading bulk carriers, as the reach of the boom is not sufficient to cover all hatches. In many cases the loading plan is such that No. 1 and No. 6 are to be loaded, which means moving the vessel under the boom in some cases two or three times.

Dual quadrant loaders are used frequently for loading large bulk carriers and can under some circumstances be more efficient and cost less than other types of shiploaders;

(e) Parallel travelling, slewing, luffing shiploader (figure 99). The luffing, slewing, travelling shiploader has the versatility required for loading large bulk carriers at high tonnage rates and in most cases provides the most economical solutio

One of the advantages of the slewing feature is that the parking position of the boom can be on the dock so that the structure is not subjected to severe wind loading. Another advantage is for ease of maintenance working directly off the dock instead of on a boom suspended over the water;

(f) Slurry loading. Slurry shiploading is a new concept and at the present time experience is limited to a narrow range of materials which can be slurried and pumped successfully. The state of the art is developing rapidly, and a number of vessels are now equipped with pumps and jets for Marconaflo discharge which will

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Table 20. Travelling versus fixed shiploader rates

Type of loader	Total tonnage shipped	Net rate	Gross rate	Effective (percentage)
Travelling	1 310 455	1 145	566	49
Quadrant	2 670 000	1 292	750	58
Fixed	985 135	915	635	70

Note. The table shows that a fixed shiploader for handling up to 500,000 tons/year can be a very efficient loader. The example used is working for about half the year under adverse weather conditions making hatch moves difficult. The other two examples are working under more moderate conditions but are plagued with ship layovers which seem difficult to control.

The net rate on the travelling and quadrant loaders is 75 per cent, whereas the fixed loader averages 61 per cent.

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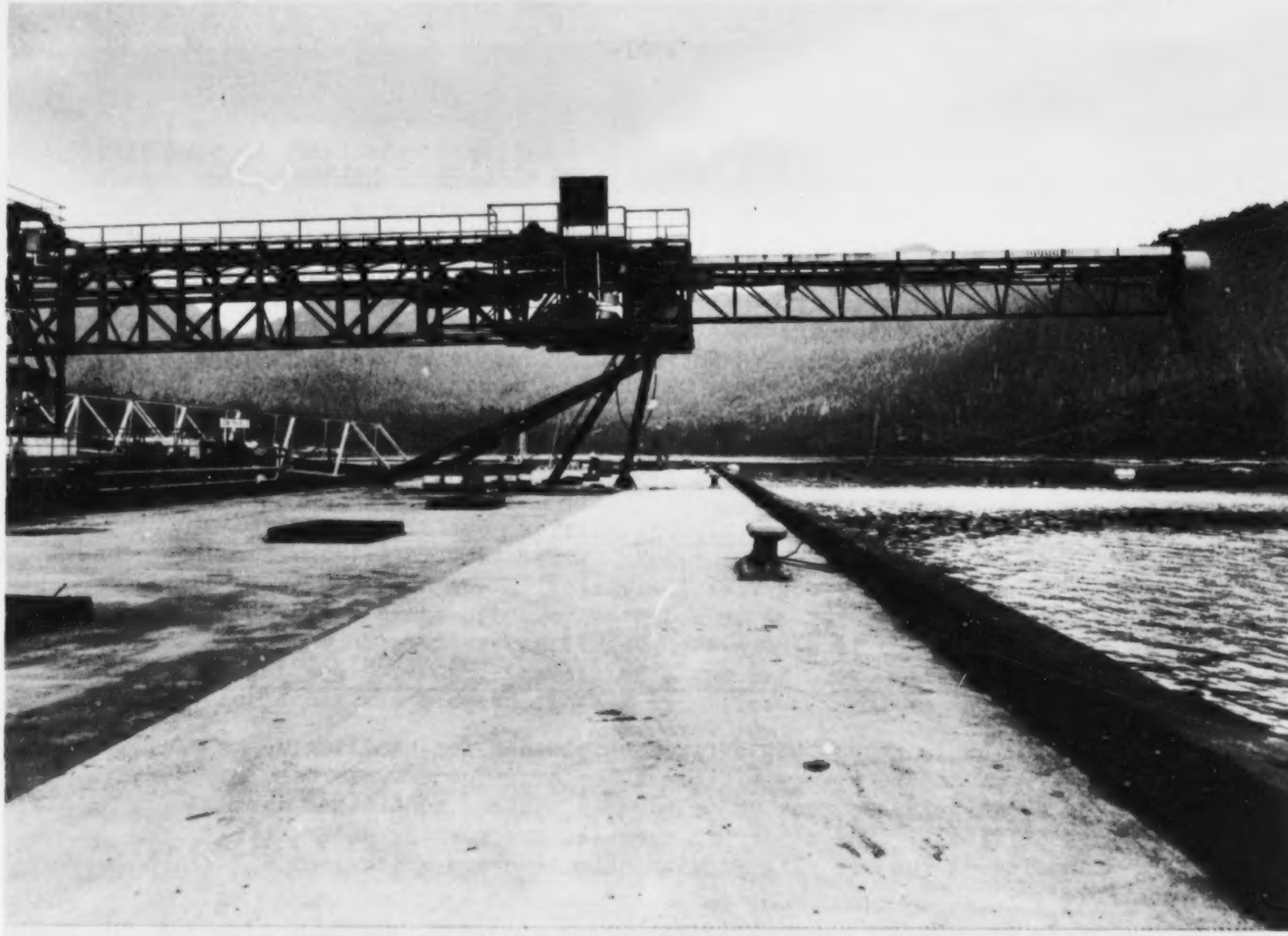
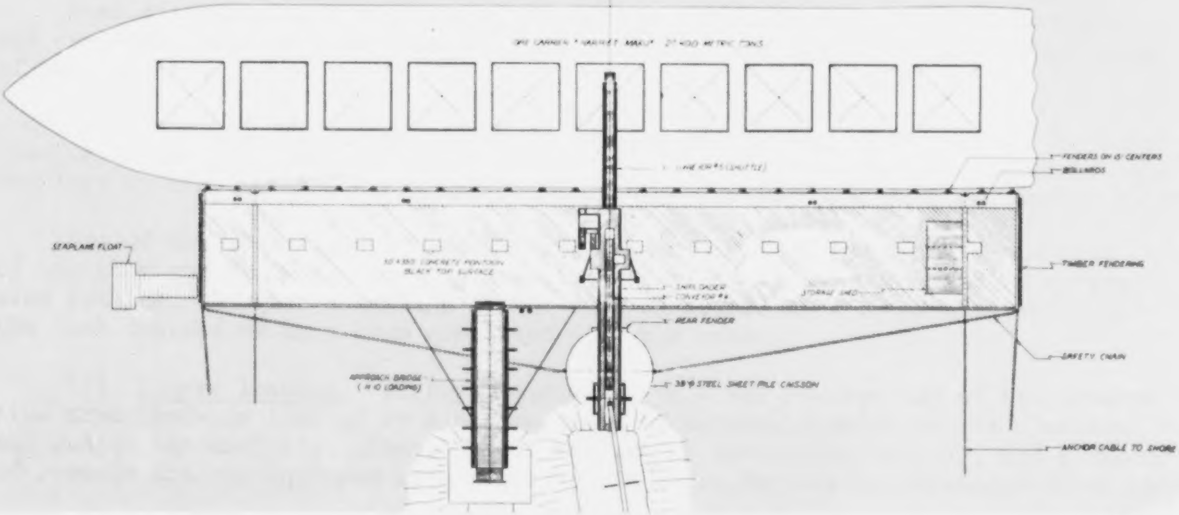


Figure 98. Concrete Pontoon and Shiploader
JEDWAY IRON ORE LIMITED
Harriet Harbour, Queen Charlotte Islands, B.C.


ORE CARRIER "HARRIET MARU" 21 400 METRIC TONS
CONVEYOR #5 (SHUTTLE)
FENDERS ON 15' CENTERS
BOLLARDS
SEAPLANE FLOAT
30 x 350 CONCRETE PONTOON
BLACK TOP SURFACE
TIMBER FENDERING
SHIPLOADER
CONVEYOR #4
STORAGE SHED
REAR FENDER
SAFETY CHAIN
APPROACH BRIDGE
(H 10 LOADING)
38'Ø STEEL SHEET PILE CAISSON
ANCHOR CABLE TO SHORE

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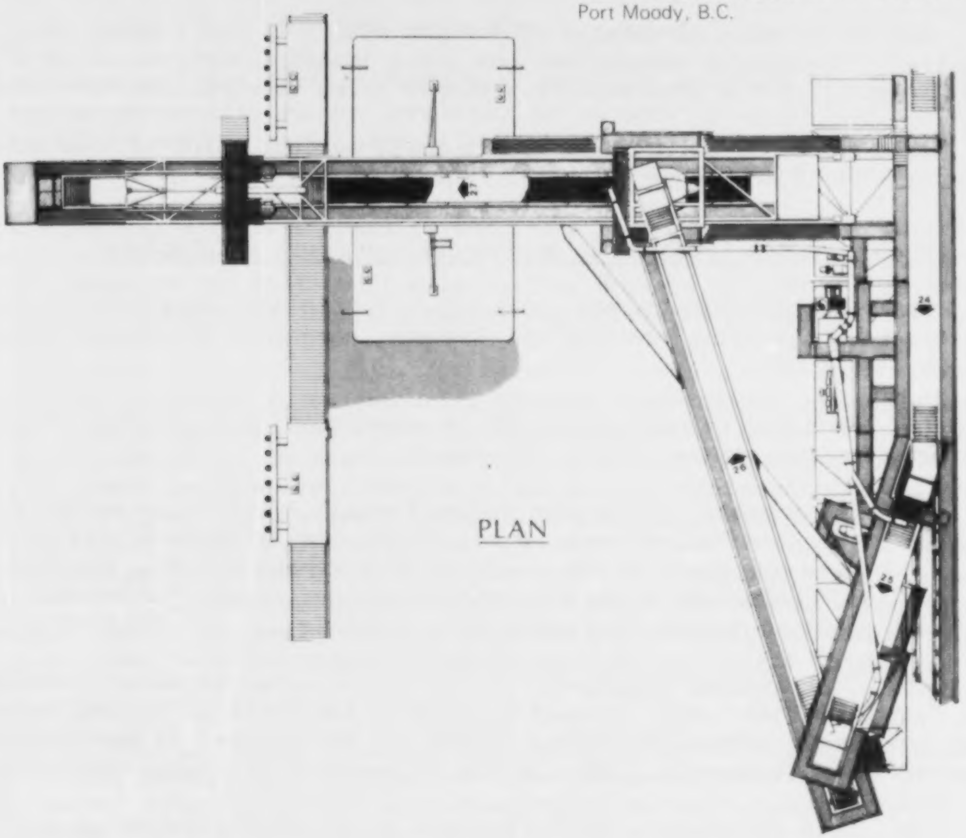

Pontoon Supported Travelling Shiploader
PACIFIC COAST BULK TERMINALS LIMITED
Port Moody, B.C.

PLAN
27
24
26
25

Figure 99

encourage this method of shiploading. The basic criteria in evaluating a loading system using a slurry method is the compatability of the commodity with slurry pumping, the dewatering characteristics and the non-deleterious effect of water on the product.

One of the main advantages of loading slurry is the minimum dock facility required. Marconaflo vessels are now being loaded almost two miles off shore in New Zealand with the ship tied to a single point mooring buoy.

This method of shiploading from a total systems point of view should be carefully considered when evaluating methods.

2. Dual parallel travelling versus dual quadrant shiploaders

Dual parallel shiploaders, although a little more expensive including substructure than dual quadrant shiploaders, have some advantages which should not be overlooked:

(a) When one shiploader is out of commission, the parallel shiploader can complete loading without moving the vessel:

(b) One parallel shiploader can be installed initially and as the annual tonnage of the terminal expands, a second parallel shiploader can be installed. This of course can apply to the quadrant arrangement as well, however. The one parallel shiploader can cover all hatches without moving the vessel, whereas with one quadrant loader this may not be possible;

(c) The parallel shiploader system can be extended along the dock face in the future to load longer vessels or service two berths. In some cases this has very noticeable advantages as one loader can be assigned to each berth, loading two ships simultaneously. This is the ultimate in effective use of equipment;

(d) Both shiploaders can be brought to a common service apron.

3. Single parallel travelling shiploader versus dual quadrant shiploaders

For an annual throughput below the point where ships begin to queue, a single, parallel, travelling shiploader has some advantage over dual quadrant loaders:

(a) The capital cost for a single shiploader is generally lower, including the runway costs. The two quadrant shiploaders also require a large elaborate support structure. The shiploader and support structure must therefore be considered when comparing costs;

(b) The operating costs for a single shiploader is half that of dual quadrant loaders as an operator is required for each unit;

/...

(c) Service and maintenance costs are for one shiploading unit only;

(d) A single parallel shiploader may be extended along the dock face to load longer vessels or service two vessels.

G. Ship unloading

The receiving and distribution facilities for bulk materials has not kept pace with the large-capacity export terminals and advance in large bulk carrier design. There is a need in some cases to upgrade facilities to receive a country's requirements integrated with an internal distribution system.

Recently several off-shore transfer terminals have been designed and several others placed in operation. These regional deep-water ports receive very large shipments of commodity for distribution in smaller parcels to ports where water depth, unloading capacity, and storage facilities would not permit delivery in supercarriers.

The shipping industry, in endeavouring to make supervessels versatile, has not developed high-speed self-unloaders for universal trade, hence the majority of unloading is carried out by shore-based gear. An exçeption exists on the Great Lakes in North America where captive trade allows the use of specialized equipment onboard ship. In the early 1900s self-unloading vessels plied the Great Lakes and from these early beginnings a great fleet of vessels has emerged. The ship sizes range from 18,000 dwt to 30,000 dwt with unloading rates of 4,000-6,000 tons/hour. Many mechanical handling systems are employed on the various ships. Anyone contemplating the construction of a self-unloader would find the Great Lakes shippers' years of experience and up-to-date technology invaluable.

A number of vessels in the international trade of the 30-50,000 dwt class are equipped with deck gantrys and clam buckets interchangeable with lumber or paper loading equipment. These ships are self-discharging to shore-based conveyor receiving systems.

Shore-based unloading systems are generally limited in capacity to 3,000 tons/hour. The clam-shell method is most commonly used and equipment is generally designed for various unloading rates and ship sizes.

1. Clam-shell unloader types

(a) Slewing boom. The slewing boom is the simplest of the clam-shell unloaders and is generally used in small operations where bucket size and unloading rate is not critical;

(b) Boom trolley, suspended clam. This machine is not found in newer installations because of the serious disadvantage of the trolley having to travel on the boom. The ratio of trolley and bucket load to payload is 100:15, causing the boom and structure to be very heavy and costly for the machine's function;

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(c) Tower hoist, boom bucket. This machine is the most efficient of the bucket unloaders. In spite of the complicated rope reaving designs and high rope maintenance these unloaders are the least expensive to purchase and have proven to be the fastest and most economical to operate.

2. Unloader capacity

Unloader capacity is limited as follows:

(a) The bucket size is limited mainly owing to the difficulty of inserting the bucket through the hatch opening onboard ship;

(b) The bucket weight is a consideration owing to the ratio of payload to bucket weight, which increases rapidly as the size enlarges;

(c) The hoisting-line size is limited to 1 5/8 or 1 3/4 inches. The stiffness of ropes beyond this diameter is such that high rope speed, combined with the number of cycles, requires very large diameter sheaves, resulting in further cost and increase in mechnical and structural costs;

(d) Automation of a clam-shell unloader is very difficult due to the variation of the digging point both vertically and horizontally. This machine can be semi-automated but must have an operator for efficiency and safety.

3. Unloading rates

In addition to the physical restrictions of unloader capacity there are other factors which influence the unloading rates.

(a) Cycle time. The clam-shell reclaimer is a high cycling machine which is operator-oriented and largely depends on the ability and resistance to fatigue of the operator;

(b) Digging rate. The free digging rate of the clam-shell should be equated to the conveyor system rate but it by no means reflects the average unloading rate. There is a wide variation in efficiency factors on digging rate which varies with ship size. Statistically the 35,000-65,000 dwt ship can achieve 50-55 per cent, while ships of 65,000-150,000 dwt only achieve 40-45 per cent. The reason for the lower efficiency on the large vessels is the lift and travel time of the bucket between digging point and the discharge hopper. In addition to this efficiency, clean-up has to be taken into consideration which in the case of a high-valued product can mean an extended time at the dock side. This makes the over-all efficiency seem very low when compared to system rate.

4. Other unloading methods

(a) Continuous bucket ladder. Well-known designers and builders of large specialized materials handling equipment in Pittsburgh, Pennsylvania, have

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developed a shore-based continuous bucket ladder unloader. This boom-suspended ladder unloader has the potential for high capacity discharge of deep-sea vessels and can achieve rates of between 4,000 and 6,000 tons/hour;

(b) Cover belt system. A Canadian materials-handling machinery company located in Belleville, Ontario, has developed a patented cover-belt system which can move materials horizontally and vertically through a loop. This special conveyor is suitable for self-unloading vessels but is somewhat limited in the materials it can handle;

(c) Onboard ship reclaimer. Several types of onboard ship reclaimers have been developed, one being a bridge-type bucket wheel, the other a two-section screw conveyor, eight feet in diameter. These reclaimers discharge to a belt conveyor running the length of the ship. Many methods are used to elevate the material to deck level and onto a discharge conveyor boom linking the ship to the shore-side handling system.

(d) Marconaflo system. Marconaflo was developed by the Marcona Corporation for unloading ship by slurry pumping. The first applications have been in unloading fine iron-ore in conjunction with their own mining operations.

The same principles apply to Marconaflo as in slurry loading from a materials-handling viewpoint. The ship, on arrival at the discharge port, connects up two hoses: one for the slurry and one for water supply, unless seawater is compatible with the commodity. The key to slurrying the compacted cargo is dependent on the high pressure "Marconajets", which through a patented nozzle deliver a high pressure stream of water and at the same time rotate slowly. The slurried cargo then flows to sumps located at each jet and is transported to shore by the ship's slurry pumping system. Unloading rates of 6,000 tons/hour can be achieved and economically justified on large vessels. This system of unloading has great potential for future bulk terminals.

H. Maintenance

Maintenance of a bulk terminal is fundamental to its successful operation as it is in the middle between land and sea transport and must not become a bottle-neck. A bulk terminal is always under pressure for maximum throughput when unloading trains or loading ships.

Machinery which is continually called on to work at maximum rates must be well maintained. It is therefore important that the terminal have a preventive maintenance programme followed closely by competent tradesmen.

Good housekeeping around a terminal is vital in terms of extended machinery life. Dust, corrosion and fire hazard are all real enemies. Special attention should be paid to electrical equipment inspection as many costly delays have been the result of electrical failure. Where large complex machinery is employed, an electrical failure can often cause serious mechanical damage or breakdown.

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Attention to mechanical detail in the replacement of scrapers, skirting, brake lining, springs, cables, and the multitude of small items found in a terminal will help ensure more available operating hours for the system. The frequent changing of filters, hydraulic oil, lube oil and adherence to machinery lubrication instructions will not only extend equipment life but reduce down-time.

I. Economics

The economics of a bulk materials handling system can only be determined by careful planning, taking into consideration the total systems approach and studying in detail the various methods of material flow. After all the parameters are defined, a concept derived and a system optimized, the capital operating and maintenance costs can be determined and related to tons handled.

The range of system costs are dependent on many factors, some of which are: the type and value of the commodity and the amount of dust control and clean-up required; the annual tonnage of commodity and the total predicted quantity; the geographical location with respect to weather conditions, availability and cost of labour, cost of land or lease and taxes.

Very large capacity systems can receive, store and ship bulk for $0.75 to $1.00 per ton where small tonnages of a difficult-to-handle commodity can cost as much as $3.50 to $4.00 per ton.

When considering the economics of the system, the direct operating cost appears to be very low per unit handled. The overheads, maintenance and efficiency of the system usually represent a large part of the handling charge. In a shipping terminal, the capital invested for intermittent operation and the efficiency of manpower represents a high unit cost as compared to a production plant which runs with higher availability of manpower and equipment.

Taking these factors into account the breakdown of operating costs, maintenance and capital recovery does not follow the normal financial analysis for an operating plant. It can be determined from the number of critical areas pointed out in this paper that the initial planning stage is very important to the economics of the system. The optimum selection of equipment to provide effective capacity and the use of reliable machines in a sequence of operations will ensure the highest possible utilization.

If these principles are followed, a competitive handling cost will result.

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38505 — April 1975 — 2100